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DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY

EXECUTIVE SUMMARY

Witteck Project 5209

for

Department of Supply and Services
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Phase III - IIC1
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MICROMEDIA

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EXECUTIVE SUMMARY

INTRODUCTION

A feasibility study was undertaken to investigate the metallurgical amenability and economic viability of a low grade gold ore from the Yukon Territory using heap leach technology. The sample used for the project was obtained from the Antoniuk Deposit which is located in Dawson Mountain Range in the West Central Yukon.

Although heap leach technology has been extensively applied to the oxide type gold and silver ores in the Western U.S.A., the technology has only recently been applied to the more competent ores prevalent in Canada. During 1985 and 1986, several test heap leach operations were successfully completed in Eastern Canada which led to decisions to implement full scale operations at those sites. The objective of this feasibility study was to determine whether a porphyry type gold ore from the Yukon Territory was amenable to heap leaching techniques and to determine the restraints imposed on the process by the climate and other environmental considerations. The results of the testing showed satisfactory gold recoveries could be achieved by heap leaching operations and that the climate would allow in excess of 200 days leaching per year. Development of capital and operating costs during the engineering feasibility phase of the project showed that the economics of the process were such as to be not favourable under the originally designated design criteria. A number of recommendations to improve the project economics are made and are summarized as follows:

- Increase the ore reserves to allow economies of scale for mining and processing

- Reduce mining costs by closer integration of heap leach site and the active mining area

- Reduce heap leach operating costs by further investigation of the reasons for high cyanide consumption on the ore

- Confirm higher heap leach recoveries that were obtained during pilot plant testing

ECONOMIC EVALUATION

The use of heap leach techniques to treat the low grade gold ore from the Antoniuk deposit was not economically attractive using the original design parameters which are summarized as follows:

Tons leached	:	743,200 tonnes per year
Head Grade	:	0.99 Au grams per tonne (0.028 oz/short ton)
Operating Cost	:	Cdn. $14.34 per tonne
Leach Recovery	:	75%
Gold Price	:	U.S. $400 per oz
Exchange Rate	:	Cdn. $1.00 = U.S. $0.72

Since the original report was written, the gold price has increased and has remained in the U.S. $450 - $460 per ounce range. At this gold price, the project economics show that the operating cost for the proposed operation will be covered by revenue from the gold produced, but does not payback the original capital investment. An increase in gold price to above U.S. $500 per ounce is required to realize a positive rate of return for the project. As shown in Table 1, the economics of the project are strongly dependent on the price of gold.

Table 1

SENSITIVITY OF PROJECT ECONOMICS TO GOLD PRICE

	Gold Price ($ US)	Internal Rate Return	Net Present Value ($Can)
	300	-128.0%	-17,940,820
	350	-112.5%	-14,004,748
Design	400	-91.3%	-10,068,675
Criteria	450	-57.7%	- 6,132,603
Cost	500	-9.2%	- 2,196,531
	600	53.0%	3,306,601

Operating costs at the property were estimated at Cdn. $14.34 per tonne of ore leached. The operating cost is substantially higher than in Nevada where operating costs are in the order of Cdn. $5-10 per tonne. The higher operating cost was the result of high cyanide consumptions in the leaching process, high ore haulage cost due to the heap leach location and the relatively small scale of the operation. As shown in Table 2 the project will show a positive internal rate of return at between Cdn. $10 and $12 per tonne of ore leached.

Table 2

SENSITIVITY OF PROJECT ECONOMICS TO OPERATING COST

	Operating Cost $/tonne	Internal Rate Return	Net Present Value ($ Can)
	10.00	19.6%	294,226
	12.00	-48.2%	-4,185,499
Design →	14.34	-91.3%	-10,068,675
Criteria	16.00	-105.6%	-14,242,210
Cost	18.00	-117.5%	-19,270,566

The ore grade of 0.99 grams per tonne during the first 3.5 years of operation was a major factor in the unfavourable economics of the proposed operation. Sensitivity analysis shown in Table 3 indicates that a head grade of between 1.4 and 1.5 grams per tonne is required during the first 3.5 years of operation to achieve a positive internal rate of return for the project.

Table 3

SENSITIVITY OF PROJECT ECONOMICS TO HEAD GRADE

	Head Grade Gram/Tonne	Internal Rate Return	Net Present Value
	0.8	-91.6%	-14,202,874
	0.9	-91.4%	-12,026,980
Design →	1	-91.3%	-9,851,086
Criteria	1.1	-91.1%	-7,675,192
Head Grade	1.2	-90.9%	-5,499,298
	1.3	-90.7%	-3,265,510
	1.4	-90.6%	-1,818,655
	1.5	24.6%	333,571
	1.6	56.9%	1,736,230

For the feasibility design criteria, gold recovery in the Antoniuk material was set at 75%. This may be a conservative estimate of the actual heap leach recovery. There were indications during the pilot plant column tests that higher extractions were possible. Actual plant practice has shown higher extractions than for pilot plant work. However, as shown in Table 4, higher gold recovery does not show major potential for improving economics of the project.

Table 4

SENSITIVITY OF PROJECT ECONOMICS TO LEACH RECOVERY

	Gold Recovery %	Internal Rate Return	Net Present Value ($ Can)
	70	-106.2%	-12,315,628
Design Criteria Recovery →	75	-91.3%	-10,068,675
	80	-71.6%	-7821722
	85	-45.8%	-5574769

MINERALOGICAL DESCRIPTION

The Antoniuk deposit can be described as a porphyry gold deposit. It comprises a heterolithic breccia body transected by rhyolite porphyry dykes and small scale structural dislocations. The breccia zone exhibits various degrees of fracture due to faults, shears and secondary infilling by replacement minerals.

Mineralization of the Antoniuk breccia comprised of slightly argentiferous gold occurring as coarse free gold. The gold is locally associated as inclusions within secondary vein system after the sulphide mineralization. Many additional minerals characteristic of epithermal-porphyry systems were observed during the mineralogical examination.

The dye penetration testwork indicated weak to excellent fracture controlled permeability, depending upon the sample under investigation. The degree and type of permeability was highly variable from sample to sample. This variability was controlled to a major extent by the degree of silicification, sericitization and oxidation and the degree of brecciation and hydrofracturing of the sample. A photograph of a typical dye penetration test on a sample from TR 21 N is shown in Photograph 1.

The gold principally occurred as coarse 50 to 300 micron free particles, but with lesser associates with iron oxyhydroxides, silicates and sulphides. The presence of base metal sulphides including sphalerite, chalcopyrite and molybdenite may cause increased cyanide consumption, and treatment of the solution recycled to the heaps and the final effluent to prevent a build-up of deleterious metals in the system may be required. A typical gold grain and its mineral association is shown in Photograph 2.

METALLURGICAL TESTWORK

Metallurgical testwork was undertaken for the Antoniuk deposit by Witteck Development in two phases. The first phase consisted of testwork performed on samples from surface trenches in the deposit and were designated TR20 and TR24. The samples were subjected to agitated leach tests to determine the optimum leaching conditions. These agitated leach tests, which are the indication of the ultimate gold extraction, showed recoveries of 99% and 88% respectively on samples TR20N and TR24N. Large scale pilot plant tests on the one inch material confirmed that high recoveries (+95%) of gold were possible using the heap leach technique on the TR20 sample. However, results on the TR24 sample showed extractions between 38% and 50% and that further metallurgical work was required to determine the reason for these lower recoveries.

In Phase II testwork, the sample T21N from the Antoniuk deposit was designated for testing as it was the sample believed to be most representative of the deposit. However, the sample may not be representative of the entire deposit as it was not located in a section of the better mineralized zones of the deposit. In addition to testwork completed on the T21N sample, testing was also carried out on the diamond drill core from the sulphide ore underlying the oxidized ore in the deposit. Summarizing the important factors from the metallurgical report, the column heap leaching tests indicated the following:

1. The final extraction of gold from Sample TR21N was independent of the feed size of the material. However, longer leach times of up to 90 days were required for coarser material to attain the final level of extraction.

2. Gold extractions varied from 68% to 87% for the Antoniuk TR21N samples tested.

3. Cyanide consumptions ranged from 3.4 to 11.3 kg/tonne for the TR21N sample. Higher lime additions of 3.9 kg/tonne of ore indicated a reduction in cyanide consumption to a more economical 2.0 kg/tonne for a sample of material crushed to passing 1 1/2".

4. Gold extractions on the drill core samples from the sulphide section of the deposit varied from 28% to 60% and with cyanide consumptions in the order of 3.5 to 5.0 kg/tonne. More extensive testwork is required to optimize gold extractions from this ore type.

5. The average head of the Antoniuk ore tested was 0.77 g/tonne. This was significantly lower than the average of the ore reserves delineated by subsequent diamond drilling which averaged 0.99 g/tonne. Although special care was taken to minimize the "nugget" effect during sampling procedures, deviations in head assays were evident. For metallurgical evaluation purposes, recoveries were calculated on average head assays and residue assays for each test.

CLIMATIC CONDITIONS

The climatic conditions that have the greatest significance on the design, construction and operation of a heap leach in the Yukon are low temperatures, wind, and precipitation.

Heap leaching cannot be undertaken during the extreme cold weather periods of the year because the cyanide solution freezes within the heap, in pumps, tanks and ponds. New techniques have been developed, however, that have allowed some operations to continue though atmospheric temperatures drop as low as -20°C.

Examination of climatic data for the area in which the proposed heap leach operation is located indicates that the daily mean temperature averages above 0°C for six months of the year, April through September. The calculated freeze/thaw index shows a net thaw for April and May which allows operations to begin so long as solutions are kept moving. The freeze/thaw index and number of leaching days is shown in Table 5.

Table 5

CLIMATOLOGICAL DATA FOR CARMACKS, YUKON
NUMBER OF HEAP LEACHING DAYS AND FREEZE/THAW INDEX

Month	Days per month	Freeze/Thaw Index	Available Leaching Days
Jan.	31	9	0
Feb.	28	7	0
Mar.	31	5	0
Apr.	30	0	30
May	31	0	31
June	30	0	30
July	31	0	31
Aug.	31	0	31
Sept.	30	0	30
Oct.	31	0	31
Nov.	30	4	0
Dec.	31	7	0
Total	365		214

It is proposed that for commercial operation, the operating season for the property would be for the seven months from April to October. Further extension of the leaching season in to November may be possible by directly injecting the leach solutions into the heap, rather than by the conventional above surface sprinkling system.

Total precipitation for the area was 269 mm, with the heaviest precipitation occurring in July and August. During July, which is the month with heaviest precipitation, it would be necessary to treat effluent at the average rate of 11 USGPM to maintain solution inventory. This calculation assumes no losses from the collection area due to evaporation from the sprinkling on the heaps and from the exposed surface of the barren and pregnant solution ponds.

For comparative purposes, annual precipitation rates in the Nevada area are in the order of 200 to 410 mm.

CAPITAL AND OPERATING COST

The capital and operating cost estimate was prepared based on the available geological data, laboratory testwork completed to date and conceptual flowsheets and plant layout drawings. The estimate was based on mining and leaching the material at a 743,200 tonnes per year rate.

Total capital cost for bringing the property into production was estimated at Cdn $2,650,000, which included a 10% contingency and engineering and construction supervision. In order to reduce the initial capital cost, it was assumed that the mining, crushing and stacking the ore on the pad was done by a contractor.

Operating costs for the property were estimated at Cdn $14.34 per tonne which included administration and head office overhead costs. The breakdown of operating costs is shown in Table 6.

Table 6

SUMMARY OF OPERATING COSTS

	Cost per Tonne of Ore Hauled	Cost per Year $
Mining ore	$ 3.70	2,749,800
Mining waste	0.71	526,500
Crushing	1.85	1,374,900
Leaching cost	7.80	5,797,600
Administration cost	0.18	128,800
Overhead	0.10	75,000
	$14.34	10,652,600

CONCLUSIONS

Although the initial design criteria indicated that the development of the Antoniuk Deposit is not economically viable at the present time, there are a number of opportunities for improving the economics of the project. When taking into consideration the limited ore reserves, the consequent treatment rate based on these reserves, and the low gold content of material, the study has shown considerable potential for the application of the heap leach process to Yukon mineral deposits. A review of the climatological data indicates that in excess of 200 days are available for leaching, which is similar to the length of the operating season in North-Eastern Ontario and Montana where heap leaches have been successfully applied. A demonstration heap leach project to prove the viability of the process in the area should be considered.

Specific recommendations for improving the economic viability of the Antoniuk Deposit have been made which include verification of higher gold recoveries, reductions in cyanide consumption, a review of the mining costs, and the scale of operation for the property. Now that more extensive drilling and a better understanding of the geology and mineral associations are available, further testing of selected samples from the Antoniuk Deposit should be completed to verify the design criteria on which the economics and recommendations made in this report are based.

SOMMAIRE

INTRODUCTION

On a entrepris une étude de factibilité pour étudier les possibilités de traitement métallurgique et la viabilité économique d'un minerai aurifère à basse teneur du Yukon, avec emploi de la technologie de lixiviation en tas. L'échantillon utilisé pour le projet provenait du gisement Antoniuk situé dans la chaîne montagneuse de Dawson, dans la région centre-ouest du Yukon.

Bien que la technologie de lixiviation en tas ait été considérablement utilisée pour le traitement de minerais contenant des oxydes d'or et d'argent dans l'ouest des États-Unis, ce n'est que récemment que cette technologie a été appliquée au traitement de minerais canadiens. En 1985 et 1986, plusieurs tests de lixiviation en tas ont été exécutés avec succès dans l'est du Canada, et ceci a conduit à la décision de réalisation d'installations commerciales sur les sites concernés. L'objectif de la présente étude de factibilité était de déterminer s'il était possible de traiter un minerai aurifère de type porphyre du Yukon en employant des techniques de lixiviation en tas et de déterminer les limites imposées au procédé par le climat et les autres facteurs associés à l'environnement. Les résultats des essais indiquent qu'il est possible d'obtenir un recouvrement satisfaisant de l'or par lixiviation en tas et que le climat permettrait d'effectuer plus de 200 jours de lixiviation par an. Le calcul de l'investissement et des frais d'exploitation au cours de la phase d'étude de factibilité technique indique que les résultats économiques du procédé ne seraient pas favorables compte tenu des critères de conception originaux. On soumet plusieurs recommandations destinées à améliorer les résultats économiques du projet, résumées comme suit:

- Augmentation des réserves de minerai pour réaliser des économies d'échelle pour l'extraction et le traitement du minerai

- Réduction des frais d'extraction par intégration plus étroite du site de lixiviation et de la zone d'extraction minière

- Réduction des frais d'exploitation de l'opération de lixiviation grace à une étude plus approfondie de la consommation élevée de cyanure par le minerai

- Confirmation des taux de recouvrement plus élevés obtenus lors des essais en installation pilote

ÉVALUATION ÉCONOMIQUE

L'emploi de techniques de lixiviation en tas pour le traitement d'un minerai aurifère à basse teneur provenant du gisement d'Antoniuk n'est pas économiquement attrayant lorsqu'on utilise les paramètres de conception originaux résumés ci-dessous:

Tonnage de minerai traité : 743 200 tonnes par an

Teneur : 0,99 gramme d'or par tonne (0,028 oz par tonne courte)

Frais d'exploitation : 14,34 $CAN par tonne

Taux de recouvrement par lixiviation : 75%

Cours de l'or : 400 $ U.S. l'once

Taux de change : 1 $CAN = 0,72 $ US

Depuis la rédaction du rapport original, le cours de l'or a augmenté et est demeuré au niveau de 450 - 460 $ US par once. À ce cours, le calcul économique indique que les frais d'exploitation de l'unité proposée seront couverts par la recette de vente de l'or produit, mais que l'investissement initial ne sera pas rémunéré. Une augmentation du cours de l'or à plus de 500 $ par once est nécessaire pour qu'on puisse obtenir un taux de rendement positif pour le projet. Comme on le voit à la table 1, les résultats économiques du projet dépendent fortement du cours de l'or.

Table 1

VARIATION DU RENDEMENT DU PROJET EN FONCTION DU COURS DE L'OR

	Cours de l'or ($ US)	Taux de rendement interne	Valeur actuelle nette ($ CAN)
	300	-128,0%	-17 940 820
	350	-112,5%	-14 004 748
Cours de l'or, selon les critères de conception ➤	400	-91,3%	-10 068 675
	450	-57,7%	- 6 132 603
	500	-9,2%	- 2 196 531
	600	53,0%	3 306 601

On a estimé les frais d'exploitation au site à 14,34 $CAN par tonne de minerai traité par lixiviation. Le coût d'exploitation est substantiellement plus élevé qu'au Nevada, où il est de 5 à 10 $ CAN par tonne. L'augmentation du coût d'exploitation est imputable à la consommation de cyanure élevée lors du traitement de lixiviation, aux frais de camionnage importants dus à l'éloignement du site de lixiviation, et à la relativement faible échelle de l'ensemble de l'exploitation. Comme on le voit à la table 2, le taux de rendement interne du projet sera positif si on peut maintenir les frais d'exploitation entre 10 et 12 $ par tonne de minerai traité par lixiviation.

Table 2

VARIATION DU RENDEMENT DU PROJET EN FONCTION DES FRAIS D'EXPLOITATION

	Frais d'exploitation $/tonne	Taux de rendement interne	Valeur actuelle nette ($ CAN)
	10,00	19,6%	294 226
	12,00	-48,2%	-4 185 499
Coût, selon les critères de conception →	14,34	-91,3%	-10 068 675
	16,00	-105,6%	-14 242 210
	18,00	-117,5%	-19 270 566

La teneur du minerai, 0,99 gramme par tonne, au cours de la première période d'exploitation (3,5 années) a été l'une des principales causes des prévisions défavorables pour l'exploitation proposée. L'analyse de sensibilité présentée à la table 3 indique qu'il faudrait que le minerai ait une teneur de 1,4 et 1,5 gramme par tonne au cours de la période initiale de 3,5 ans pour que le projet ait un taux de rendement interne positif.

Table 3

VARIATION DU RENDEMENT ÉCONOMIQUE DU PROJET EN FONCTION DE LA TENEUR DU MINERAI

	Teneur du minerai gramme/tonne	Taux de rendement interne	Valeur actuelle nette ($ CAN)
	0,8	-91,6%	-14 202 874
	0,9	-91,4%	-12 026 980
Teneur, selon les critères de conception →	1	-91,3%	-9 851 086
	1,1	-91,1%	-7 675 192
	1,2	-90,9%	-5 499 298
	1,3	-90,7%	-3 265 510
	1,4	-90,6%	-1 818 655
	1,5	24,6%	333 571
	1,6	56,9%	1 736 230

Dans le cadre de l'étude de factibilité, on a fixé le taux de recouvrement de l'or du minerai d'Antoniuk à 75%. Ceci peut être une estimation conservatrice du taux de recouvrement réel. Les résultats des essais en colonne (installation pilote) indiquent qu'il est possible d'obtenir un taux d'extraction plus élevé. Les résultats réels en usine indiquent des taux d'extraction plus élevés que pour les essais en installation pilote. Cependant, les résultats présentés à la table 4 indiquent que l'augmentation du taux de recouvrement de l'or ne permet pas de rentabiliser le projet.

Table 4

VARIATION DU RENDEMENT ÉCONOMIQUE DU PROJET EN FONCTION DU TAUX DE RECOUVREMENT

	Taux de recouvrement de l'or %	Taux de rendement interne	Valeur actuelle nette ($ CAN)
	70	-106,2%	-12 315 628
Taux de recouvrement selon les critères de conception	→ 75	-91,3%	-10 068 675
	80	-71,6%	-7821722
	85	-45,8%	-5574769

MINÉRALOGIE

On peut décrire le minerai du gisement Antoniuk comme un minerai de porphyre aurifère. Il s'agit d'un gisement de brèche hétérolitique traversé de veines de porphyre rhyolite et de petites dislocations structurales. Dans la zone de brèche, on observe divers degrés de fracturation due aux failles, de cisaillements, et des zones secondaires comblées par des mineraux de remplacement.

La minéralisation de la brèche d'Antoniuk contient de l'or légèrement argentifère, sous forme d'or brut libre. L'or est associé localement sous forme d'inclusions au sein du système de veines secondaires après la minéralisation de sulfure. On a observé de nombreuses autres caractéristiques minérales des systèmes de porphyre épithermale au cours de l'examen minéralogique.

Les essais de pénétration par colorants ont indiqué une perméabilité faible à excellente contrôlée par la fracturation, selon l'échantillon étudié. Le degré et le type de perméabilité varient considérablement d'un échantillon à un autre. Cette variabilité est contrôlée en grande partie par le degré de silicification, séricitisation et oxydation, et le degré de bréchification et d'hydrofracture de l'échantillon. La photographie 1 présente un échantillon de TR 21N soumis à un essai typique de pénétration par colorant.

L'or est présent essentiellement sous forme de particules libres grossières de 50 à 300 microns associées à de plus petites particules contenant des silicates, sulfures et oxyhydroxydes de fer. La présence de sulfures de métaux non ferreux dont sphalérite, chalcopyrite et molybdénite peut provoquer une augmentation de la consommation de cyanure, et rend nécessaire un traitement de la solution recyclée à la pile de lixiviation et de l'effluent final pour empêcher une accumulation de métaux dangereux dans le système. On présente à la photographie 2 un grain d'or typique et les mineraux qui y sont associés.

ESSAIS MÉTALLURGIQUES

Witteck Development a entrepris le programme d'essais métallurgiques sur le minerai d'Antoniuk en deux phases. La première phase impliquait l'exécution d'essais sur des échantillons provenant de tranchées de surface du gisement (essais TR20 et TR24). Les échantillons ont été soumis à des essais de lixiviation par agitation; ceci devait permettre de définir les conditions optimales de lixiviation. Ces essais de lixiviation par agitation, qui donnent une indication du taux d'extraction final de l'or, ont produit respectivement des taux de recouvrement de 99% et 88% sur les échantillons TR20N et TR24N. Les essais en installation pilote à grande échelle sur le matériau broyé à un pouce ont confirmé qu'il était possible d'atteindre un taux de recouvrement élevé (95%) par emploi de la technique de lixiviation en tas pour l'échantillon TR20. Cependant, le taux d'extraction n'a été que de 38% et 50% pour l'échantillon TR24; on devra exécuter d'autres travaux métallurgiques pour déterminer les causes de ce taux faible.

Pour les travaux de la phase II, l'échantillon T21N du gisement Antoniuk a été testé comme s'il s'agissait de l'échantillon le plus représentatif du gisement. Cependant l'échantillon peut ne pas être représentatif de l'ensemble du gisement car il ne provient pas d'une section des meilleures zones minéralisées du gisement. En plus des essais effectués sur l'échantillon T21N, on a également effectué des tests sur une carotte de forage provenant du minerai sulfuré situé sous le minerai oxydé du gisement. On peut résumer comme suit les plus importants facteurs identifiés dans le rapport métallurgique des essais de lixiviation en colonne.

1. Le taux d'extraction final de l'or de l'échantillon TR21 a été indépendant de la taille de broyage du matériau. Cependant, on a dû traiter les matériaux grossiers pendant de plus longues périodes, jusqu'à 90 jours, pour atteindre le niveau d'extraction final.

2. Le taux d'extraction de l'or a varié de 68% à 87% pour les échantillons TR21N provenant d'Antoniuk.

3. La consommation de cyanure a varié de 3,4 à 11,3 kg/tonne pour l'échantillon TR21N. L'augmentation des additions de chaux à raison de 3,9 kg/tonne de minerai a permis de réduire la consommation de cyanure au niveau plus économique de 2,0 kg/tonne, pour un échantillon de matériau broyé à moins de 1,5 po.

4. Le taux d'extraction de l'or sur les échantillons prélevés par carottage dans la section de sulfure du gisement variait de 28 à 60%, et la consommation de cyanure était de 3,5 à 5 kg/tonne. On devra effectuer des essais plus approfondis pour optimiser le taux d'extraction de l'or à partir de ce type de minerai.

5. La teneur moyenne du minerai d'Antoniuk testé était de 0,77 g/tonne. Cette teneur était sensiblement inférieure à la teneur moyenne des réserves identifées par les travaux ultérieurs de forage-diamant, qui ont indiqué une teneur moyenne de 0,99 g/tonne. Bien qu'on ait pris soin de minimiser l'effet de pépite lors de l'échantillonnage, des déviations ont été évidentes dans les analyses du minerai. Aux fins de l'évaluation métallurgique, on a calculé les taux de recouvrement sur la base des valeurs moyennes des analyses du matériau traité et des résidus pour chaque test.

CONDITIONS CLIMATIQUES

Basse température, vent et précipitations sont les caractéristiques climatiques qui ont l'effet le plus important sur la conception, la construction et l'exploitation d'une installation de lixiviation en tas au Yukon.

On ne peut entreprendre la lixiviation au cours de la période de froid extrême parce que la solution de cyanure se congèle au sein de la pile, dans les pompes, dans les réservoirs et dans les bassins de sédimentation. Cependant, de nouvelles techniques qui permettent de poursuivre les opérations après une baisse de la température atmosphérique jusqu'à -20°C ont été mises au point.

L'examen des données climatiques pour la région dans laquelle le site de l'installation de lixiviation proposée est situé indique que la température quotidienne moyenne est supérieure à 0°C pendant six mois de l'année, d'avril à septembre. L'indice de gel/dégel calculé indique qu'il est possible d'exploiter l'installation d'avril à mai dans la mesure où les solutions sont maintenues en mouvement. Les indices de gel/dégel et nombre de jours de lixiviation sont présentés à la table 5.

Table 5

DONNÉES CLIMATOLOGIQUES POUR CARMACKS, YUKON NOMBRE DE JOURS DE LIXIVIATION EN TAS ET INDICE DE GEL/DÉGEL

Mois	Jours par mois	Indice de gel/dégel	Nbre de jours de lixiviation disponibles
Janv.	31	9	0
Fév.	28	7	0
Mars	31	5	0
Avr.	30	0	30
Mai	31	0	31
Juin	30	0	30
Juil.	31	0	31
Août	31	0	31
Sept.	30	0	30
Oct.	31	0	31
Nov.	30	4	0
Déc.	31	7	0
Total	365		214

On propose que dans le cas de l'exploitation commerciale la saison d'exploitation comprenne sept mois, d'avril à octobre. Il pourrait être possible de prolonger la saison de lixiviation jusqu'au mois de novembre, en injectant directement les solutions de lixiviation dans la pile au lieu d'employer la méthode conventionnelle d'aspersion en surface.

Le total des précipitations dans la région est de 269 mm; les précipitations les plus importantes surviennent en juillet et en août. Au cours du mois de juillet, le mois où on enregistre les plus importantes précipitations, il serait nécessaire de traiter l'effluent au rythme moyen de 11 gal. US par minute pour maintenir le stock de solution. Dans ce calcul, on suppose qu'il n'y a pas de perte dans la zone de collecte du fait de l'évaporation à partir du système d'aspersion des tas et à partir des surfaces exposées des bassins de solution vierge et chargée d'or.

Pour comparaison, on rappelle qu'au Nevada on enregistre des précipitations annuelles de 200 à 400 mm.

INVESTISSEMENT ET FRAIS D'EXPLOITATION

On a préparé des estimations du montant de l'investissement et des frais d'exploitation sur la base des données géologiques disponibles, des travaux de laboratoire exécutés jusqu'à ce jour, et des circuits de traitement conceptuels et dessins d'implantation de l'usine. Pour ces estimations, on a choisi l'hypothèse d'un volume de traitement (extraction minière et lixiviation) de 743 200 tonnes par an.

L'investissement total nécessaire pour mettre la propriété en production est estimé à 2 650 000 $ CAN, ce qui comprend une allocation de 10% pour frais imprévus et travaux d'ingénierie et supervision de la construction. Pour réduire l'investissement initial, on a supposé que les travaux d'extraction, broyage et empilage du minerai sur la dalle de traitement seraient confiés à un entrepreneur.

Les frais d'exploitation de l'installation sont estimés à 14,34 $CAN par tonne, ce qui comprend les frais d'administration et frais généraux du siège social. La ventilation des frais d'exploitation est présentée à la table 6.

Table 6

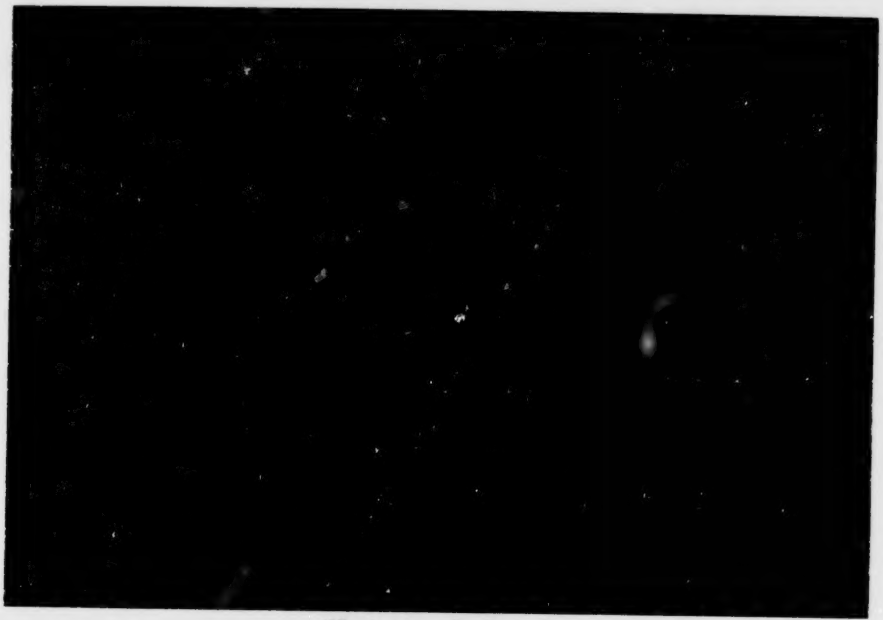
SOMMAIRE DES FRAIS D'EXPLOITATION

	Coût par tonne de minerai traité	Coût par année $
Extraction du minerai	3,70 $	2 749 800
Rebuts d'extraction	0,71	526 500
Broyage	1,85	1 374 900
Frais de lixiviation	7,80	5 797 600
Frais administratifs	0,18	128 800
Frais généraux	0,10	75 000
	14 34 $	10 652 600

CONCLUSIONS

Bien que le modèle de conception initial ait indiqué que le gisement d'Antoniuk ne serait pas économiquement viable actuellement, plusieurs possibilités d'améliorations des perspectives économiques du projet existent. Lorsqu'on considère les réserves de minerai limitées, le taux de traitement basé sur ces réserves, et la faible teneur en or du matériau, l'étude révèle un potentiel considérable pour l'application du procédé de lixiviation en tas pour les gisements minéraux du Yukon. L'examen des données climatiques indique qu'on peut disposer de plus de 200 jours de lixiviation par an, ce qui est similaire à une saison d'exploitation dans le nord-est de l'Ontario et au Montana, où le procédé de lixiviation en tas a été utilisé avec succès. On devrait considérer un projet de démonstration de lixiviation en tas pour démontrer la viabilité du procédé dans la région.

On a soumis des recommandations spécifiques pour l'amélioration de la viabilité économique du gisement Antoniuk; ceci comprend la vérification de taux de recouvrement de l'or plus élevés, la réduction de la consommation de cyanure, un examen des frais d'exploitation minière, et le choix de l'échelle d'exploitation de la propriété. Maintenant qu'on a accès aux résultats d'importants travaux de forage et qu'on comprend mieux la géologie du gisement et sa minéralisation, on devrait exécuter d'autres essais sur des échantillons choisis du gisement Antoniuk pour vérifier les critères de conception sur lesquels on a basé les calculs économiques et recommandations de ce rapport.



Photograph No. 1
Dye penetration test sample from trench TR20 after 21 days exposure. Permeability along major fractures, adjacent microfractures and minor diffuse patches within silicified mineralization.

125 microns

Photograph No. 2 - 200x Reflected Light
Coarse liberated particle of native gold exhibiting ragged margins with irregular tendrils and abundant porosity. Polished section of heavy mineral gravity concentrate.

Volume

2

REQ. NO. 6-9006

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CONTRACT REPORT NO. 69006-01-SQ

DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY
PHASE I
METALLURGICAL TEST WORK AND
MINERALOGICAL EVALUATION

for

Archer, Cathro and Associates
1016-510 West Hastings Street
Vancouver, British Columbia

Project No. [illegible]181

February 14, 1986

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1.0 SUMMARY

Metallurgical test work and mineralogical studies were performed on gold ore samples from three Yukon properties, which are currently under mineral exploration. The samples, from the Antoniuk (Mt. Freegold), Casino, and Revenue deposits, contain low gold grades and testwork was conducted to evaluate the amenability of the ores to recovery by low cost heap leach cyanidation. The test work included permeability characterization, bench scale agitated leach cyanidation and small pilot scale column heap leach cyanidation. Microscopic characterization of gold mineralization was performed on the three gold samples. Dye penetration testing showed excellent penetration along fracture surfaces within the Freegold (TR20) and Casino zones. Diffuse penetration, locally associated with fractures, was observed in the Revenue, Casino, and Freegold (TR21, TR18) zones. Diffuse matrix penetration was typically weak to moderately developed in all zones except the TR18 zone. The TR18 zone exhibited excellent matrix penetration.

The gold extraction results of the heap leach and agitated leach cyanidation tests are summarized below.

Deposit	Agitated Leach (pH 10.5) (pH 11.5) % Au extraction	Heap Leach (pH 9.9-10.5) % Au extraction
Freegold		
-TR20	99.0	95.6
	99.9	
-TR24	90.5	38.5
	86.2	
Revenue	44.9	47.1
	77.1	
Casino	71.1	23.0
	75.3	

Agitated leach cyanidation resulted in higher gold extraction than the heap leach tests. Increased leach recoveries at higher pH (11.5) were indicated in the Revenue, Casino and TR20 samples, while a slight decrease was observed in the TR24 samples. However, the higher pH values resulted in large increases in lime consumption accompanied by decreased cyanide consumption. High levels of copper were solubilized during cyanidation in the Revenue ore sample. Problems with mass balancing leaching results were encountered throughout the program. This was due to cutting head samples from 15 cm bulk samples.

The mineralogical studies indicated that much of the gold occurred as coarse free particles locally associated with gangue minerals and/or secondary iron oxidation products. High leach extraction in the TR20 samples suggest that gold mineralization was primarily fracture controlled. Low heap leach extractions in the Revenue, Casino and TR24 zones result from poor permeability and possibly disseminated gold mineralization. High levels of copper solubilized during cyanidation of the Revenue ore sample may complex with the cyanide. This can result in poor gold recovery and higher cyanide consumption.

2.0 INTRODUCTION

In first phase of the metallurgical and engineering assessment of the Dawson Range Heap Leach Project, six surface bulk samples were sent to Witteck Development Inc.. The samples were as follows:

1-4. Antoniuk (Mt. Freegold) Deposit - TR18 zone
- TR20 zone
- TR21 zone
- TR24 zone

5. Casino Deposit
6. Revenue Deposit

The test program included the following:

1. Permeability testing (Dye penetration)
2. Bench scale agitated leach cyanidation
3. Small pilot scale column heap leach cyanidation
4. Mineralogical evaluation

The purpose of the Phase I test work was to evaluate the ores, particularly the Antoniuk (Mt. Freegold) deposit, for amenability to low cost heap leach cyanidation. Bench and small pilot plant scale heap leach and bench scale agitated leach tests were utilized for assessment of cyanidation parameters. These parameters included cyanide consumption, lime consumption, pH, gold extraction, and solution fouling. The degree of variability between the three deposits will be discussed, particularly with respect to gold recovery for crushed versus uncrushed minerals. The parameters will be utilized as a data base for further metallurgical and engineering assessment. Mineralogical studies were utilized to evaluate the gold mineralization in terms of particle size, chemical composition, mineral association and variability between deposits. A brief summary of sample preparation procedures and optical and scanning electron microscopy procedures are included in the report.

3.0 PERMEABILITY TEST WORK

3.1 Procedures

One rock sample representing each of the six ores was chosen for permeability test work. Each rock was washed to remove loose debris and dried with a paper towel. Each sample was placed in a shallow pan contining 2 cm of "Zyglo", a water washable fluorescent penetrant dye.

Samples were cut at various times (1, 3, 7, 11, 15 days or 1, 4, 8, and 11 days) using a diamond tipped rock saw. The rock was rinsed with hot water and then acetone and dried to remove excess dye. The sample was placed adjacent to two fluorescent black lights. The dye fluoresces to a yellow green colouration which indicates the degree of penetration. After photographs were taken, samples were replaced, in the same orientation, in the dye for further dye penetration.

3.2 Results

Based on the test work, the following results were obtained.

3.2.1 Revenue

Photographs of this sample were taken at exposures of 1, 3, 7, and 15 days. After Day 1 (Figure 1), the dye had penetrated approximately 0.30 cm into the sample where it was submerged in the dye. After further exposure (Figure 2), the dye crept up the margins of the rock sample and displayed diffuse penetration along these areas.

There was no noticeable change in penetration from Day 7 (Figure 3) to Day 15 (Figure 4). Dye penetration reached a maximum of 0.90 cm. The core of the sample displayed no dye penetration. Fracture density of the Revenue sample was low to nil.

3.2.2 Casino

Photographs of this sample were taken after exposures of 1, 3, 5, 7 and 15 Days. After Day 1 (Figure 5), penetration was observed in most fractures and pore spaces. Diffuse penetration adjacent to fractures and margins of the rock sample were observed by Day 3 (Figure 6). Diffuse matrix penetration, to a maximum of 0.60 cm, was observed from Day 7 (Figure 7) to Day 15 (Figure 8). Fracture density of the Casino sample ranged from low to moderate.

3.2.3 Freegold TR20

Observations of this sample were performed after exposures of 1, 3, 7 and 15 Days. Excellent penetration into fractures was observed after Day 1 (Figure 9). After Day 3 (Figure 10), weak to moderate diffuse penetration adjacent to fractures was observed. This type of penetration increased slightly by Day 15 (Figures 11 and 12). Fracture density was moderate to high.

3.2.4 Freegold TR21

This sample was examined after exposures of 1, 4, 8, and 11 Days. After Day 1 (Figure 13), penetration occurred in cracks and pores located in the bottom portion of the sample where it was submerged in the dye. Penetration was typically limited to pore spaces even after prolonged exposure (Figures 14, 15, 16). Fracture density was typically low to nil.

3.2.5 Freegold TR18

This sample was examined after exposures of 1, 4, 8, and 11 Days. The first section, after Day 1 (Figure 19), showed diffuse penetration covering approximately 40 % of the freshly cut surface and progressed to 90% after 11 Days of exposure (Figures 18, 19, 20). Penetration along fractures was good, though fracture density was typically low.

3.2.6 Freegold TR24

Photographs of this section were taken at exposures of 1, 3, 7 and 15 Days. After Day 1 (Figure 21), penetration occurred only in the rare fractures. Less commonly, the dye penetrated the few pores spaces. Continued exposure (Figure 22, 23, 24) displayed no increase in degree of dye penetration.

3.3 Discussion

The dye penetration test work performed on samples from the Casino and Revenue deposits exhibited limited potential for heap leaching due to a paucity of fractures and poor dye penetration. Crushing will likely be necessary for heap leaching of the Revenue deposit. The Casino deposit, which displayed very poor dye penetration and low fracture density, would at least require fine crushing. In both deposits, the absence of fracture controlled mineralization would indicate that a reduced heap leach recovery can be expected.

The Antoniuk (Mt. Freegold) deposit varies in fracture density and degree and type of permeability from zone to zone. TR18 provided the best results with excellent diffuse penetration with associated fracture permeability. The TR20 zone exhibited strong penetration into fractures with limited diffuse penetration. TR24 and TR21 zones, with lower fracture densities, exhibited marginal diffuse penetration and weak to moderate pore and fracture penetration. The TR18 sample indicated high potential for heap leaching of coarse material. The TR20 sample would provide excellent material for heap leaching provided the gold mineralization was confined to fracture control. The TR24 and TR21 zones would require fine crushing and therefore represent poorer material for heap leaching.

4.0 MINERALOGICAL EVALUATION

4.1 Procedures

4.1.1 Preparation

The ore samples were ground by laboratory ball mill and passed through a minus 48 mesh screen (0.295 mm). The oversize was reground and the process repeated until the entire sample passed the 48 mesh screen size.

Representative samples of 6 mesh material were riffled out and prepared to pass minus 100 mesh (0.147) for head assays. All assays were completed by fire assay with direct current plasma spectroscopy finish. The results of the head assays are given in Table 1.

The ground ore samples were vanned in order to produce concentrates with higher gold content for the preparation of polished mounts. The ores were vanned on a 25 cm vanning plaque and the tails were re-vanned. The concentrates were then dried and weighed. The tailings samples were filtered, dried and assayed for gold. The grade of the concentrate was calculated by subtraction.

The sample weights, assays and calculated concentrate analysis for the vanned ores are given in Table 1.

One 25 mm polished mount was prepared for each concentrate.

Table 1

GOLD RECOVERY IN VANNED CONCENTRATES

Zone	Head Au (gpt)	Van Tails Au (gpt)	Van Conc Wt (g)	Van Tail Wt (g)	Van Conc Au (gpt) (calculated)
Casino	1.10	0.65	12.70	476	18.9
TR20	0.71	0.04	0.80	454	420
Revenue	1.15	0.25	1.60	431	292

4.1.2 Examination

The polished mounts for each ore sample were scanned at magnification of 100x, 200x, 500x and 1250x, the latter using oil immersion. Grains of specific interest were photomicrographed using a 35 mm camera attached to the Leitz Laborlux 12 reflected light microscope.

Numerous grains were pinpointed by circling using a diamond tipped marker objective for subsequent scanning electron microscopy (SEM). Prior to the SEM examination, each mount was carbon coated under vacuum. The thickness of the carbon layer was approximately 200 Angstroms. The grains of interest were located and examined at magnifications up to 5000x. An energy dispersive x-ray analysis provided a qualitative spot analysis (EDAX) of the targetted particle. The resolution of the instrument is in the order of one micron.

Mineral identifications are based upon a combination of optical properties using the petrographic microscope, secondary electron images from the scanning electron microscope, and chemical data from the EDAX CRT display. The EDAX spectra were photographed from the analysis with a Polaroid attachment. In the absence of quantitative chemical data and the presence of overlapping optical properties, tentative identification of minerals has been noted with a short list of possible minerals.

4.2 Detailed Mineralogy

The following is a detailed description of the minerals and their associations identified from the three vanned concentrates (Revenue, Casino and TR20). Photomicrographs and EDAX spectra are displayed in Appendix II.

4.2.1 Revenue

The Revenue sample was chiefly comprised of pyrite (Figure 25) with lesser arsenopyrite (Figure 28), marcasite (Figure 28), and iron-titanium oxides. The latter include magnetite and ilmenite with abundant secondary products comprising hematite, goethite and possibly lepidocrocite. Rutile and sphene are also common.

The sulfide minerals included chalcopyrite, pyrrhotite, galena, bornite, digenite, covellite, chalcopyrite and sphalerite. Rare occurrences of baryte and tellurobismuthite were observed. Fragments of malachite were common.

Gold particles formed both as free and middling particles. Other gold particles were associated with oxide crusts on pyrite.

Pyrite typically occurred as angular, massive to brecciated particles and range from less than 10 microns to 300 microns. The pyrite particles average approximately 150 microns. The pyrite rarely occurred as subhedral to euhedral (idiomorphic) grains. Pyrite commonly contained angular to elongate subrounded inclusions ranging from less than 5 microns to 50 microns. These inclusions were comprised of rutile, sphene, chalcopyrite, galena, tellurobismuthite, pyrrhotite, bornite, hematite, goethite or limonite, ilmenite and magnetite.

Pyrite grains were commonly affected by oxidation. Replacement along grain boundaries and fractures by secondary iron oxides (Figure 25) was observed. The replacement was exhibited by crustification and collomorphic layers (Figure 28). The degree of replacement varies from weak to pervasive.

Arsenopyrite typically formed angular, anhedral particles ranging in size up to 230 microns. The arsenopyrite may contain fine inclusions, of less than 10 microns in diameter, of bornite and chalcopyrite or pyrrhotite and chalcopyrite. Fracturing of the arsenopyrite ranged from weak to intense (Figure 28). Replacement, by oxidation to pyrite, was common.

Marcasite formed angular aggregates of broad laths and sector twinned grains ranging in size to 90 by 240 microns. Aggregates of fine grained marcasite ranged up in size to 200 x

350 microns and average between 100 to 150 microns in diameter. These aggregates commonly display speckled birds eye type replacement by oxides. The coarse grained marcasite (Figure 29) commonly exhibited moderate to intense fracturing, and replacement by secondary oxides (goethite, limonite) (Figure 30).

Pyrrhotite occurred as angular free particles up to 350 microns in diameter, minute inclusions within pyrite or arsenopyrite, and may form irregular, porous crusts replacing pyrite (Figure 32). Locally, the pyrrhotite particles exhibit intense brecciation and replacement by later oxides.

Chalcopyrite occurred as inclusions, up to 50 microns in diameter, within pyrite (Figure 33, 24), and free to middling particles associated with pyrite, pyrrhotite, or goethite (Figure 31). The chalcopyrite particles are rare and commonly range in size from 90 to 120 microns. The maximum diameter of chalcopyrite was approximately 200 microns. Locally the chalcopyrite exhibited replacement by bornite, covellite, digenite or chalcocite. Malachite formed coarse fragments with radiating blades and collomorphic growth zoning up to 350 microns in diameter (Figure 36.) The composition of the malachite is shown in Figure 37.

Sphalerite formed anhedral to subhedral particles ranging in size from 55 to 100 microns (Figure 38). Galena occurrred as free particles (Figure 39, 40) ranging in size up to 90 x 135 microns. The galena also occurred as minute inclusions in pyrite (Figure 26). The composition of the inclusion is shown in Figure 27. A minute inclusion of tellurbismuthite was observed within pyrite (Figure 41). The composition of the inclusion, measuring 4 by 7 microns, is shown in Figure 42. Another inclusion of tellurobismuthite was observed within pyrite and is shown in Figures 43 and 44.

Magnetite and ilmenite commonly form exsolution intergrowths and formed particles up to 350 microns in diameter. Minute blebs of chalcopyrite may be present. Myrmekitic intergrowths and homogenous oxide particles are typically replaced along cleavages by goethite and hematite. Angular to subhedral particles of rutile were common (Figure 45). The composition of the rutile particles is given in Figure 35. Subangular to subhedral baryte particles were present (Figure 46).

Gold Mineralization in Revenue Ore

Two occurrences of native gold were observed in the Revenue concentrate. The first, a minute 8 x 12 micron particle, was enclosed by a secondary oxide crust surrounding the pyrite particle (Figure 47). The composition of the particle is shown in Figure 48. The second occurred as a free particle (Figure 49, 50) exposed as a v-shaped grain 80 x 80 microns in diameter. The particle was argentiferous (see Figure 51) and displayed a partial rim of gold which contained a lesser amount of silver (Figure 52).

4.2.2 Casino

The Casino ore sample was chiefly comprised of iron-titanium oxides (Figure 53) consisting of magnetite and ilmenite with lesser hematite, goethite, rutile and sphene.

The sulfide minerals were comprised of chalcopyrite, pyrite, arsenopyrite, pyrrhotite, covellite, bornite and digenite. No gold occurrences were observed in the vanned concentrate of the Casino ore.

Magnetite and ilmenite form homogeneous particles (Figure 53), myrmekitic intergrowth, lamellar intergrowth typically with ilmenite (Figure 54) as fine ribbon to bleb type lamellae, and aggregates of rounded subhedra. The particles commonly range from 20 - 225 microns and reach 420 microns in diameter. The particles averaged 80 microns in diameter. Magnetite-ilmenite formed middling and inclusion type particles with chalcopyrite up to 150 microns in diameter. Pyrite subhedra up to 65 microns were present. Pyrrhotite occurred as angular anhedral particles up to 90 microns in diameter. Arsenopyrite typically formed angular anhedra. The particles averaged approximately 70 microns in diameter and ranged to a maximum of 90 by 200 microns. Chalcopyrite formed subrounded to subhedral inclusions in magnetite-ilmenite (Figure 55) and rarely formed free particles. The particles were typically less than 25 microns with a maximum diameter of 40 microns. Covellite locally formed middling particles, up to 65 microns in diameter. The Fe-Ti oxides exhibited peripheral, fracture, and preferential (magnetite) replacement by goethite and hematite

(Figure 56). Ilmenite displayed rim alteration to rutile and sphene. Hematite also occurred as radiating sprays of euhedral blades. Individual particles had a maximum diameter of 130 microns while aggregates attained 250 microns (Figure 53). Goethite occurred as massive to banded collomorphic crusts with cores of primary oxides and aggregate grains up to 175 microns. The goethite locally occurred as a matrix to fine grained pyrite particles.

Pyrite formed minute inclusions within magnetite-ilemite bornite, digenite and chalcopyrite. Covellite commonly formed an alteration rim on chalcopyrite inclusions (Figure 57) with iron-titanium oxides. Bornite and chalcopyrite exhibit lenticular exsolution lamellae (Figure 58). Elongate to subrounded smooth inclusions of pyrrhotite ranging from 11 to 26 microns were observed within arsenopyrite (Figure 59). The inclusions may be associated with rutile and/or sphene. The EDAX spectra of the pyrrhotite is given in Figure 60.

4.2.3 Freegold-TR20

The Freegold TR20 sample was chiefly comprised of arsenopyrite with lesser pyrite, pyrrhotite and iron-titanium oxides, primarily magnetite and ilmenite. The ore sample also contained chalcopyrite, sphalerite, baryte, rutile, sphene, goethite, limonite or lepidocrocite, bornite, chalcocite, and covellite. Minor fragments of metallic iron (Figure 64, 65) may be due to contamination during sample preparation.

The arsenopyrite typically formed large angular fragments (Figure 61) and less commonly aggregates of fine to medium grained subhedra. The angular arsenopyrite averaged approximately 115 microns in diameter and had a maximum size of 300 by 450 microns. The arsenopyrite may exhibit lamellar or penetration twinning. Inclusions of pyrrhotite up to 30 microns in diameter are common. The inclusions are typically subrounded and less than 5 - 10 microns in diameter. Arsenopyrite formed middling particles with pyrite and iron-titanium oxides. Replacement in the form of crustification, fracture filling and collomophic layering by goethite and possibly limonite affects many of the arsenopyrite fragments (Figure 63). Replacement by pyrrhotite, as peripheral crusts and collomorphic lobes, was also observed in the arsenopyrite.

Angular particles of pyrite and marcasite up to 300 microns, are commonly replaced by goethite and other oxide (Figure 62). The pyrite also exhibited replacement by collomorphic pyrrhotite. Pyrrhotite formed free angular particles up to 140 by 220 microns. Minute inclusions of pyrrhotite within pyrite and arsenopyrite average 5 microns in diameter and range up to 40 microns in size.

Galena occurred as minute inclusions (Figure 66) within arsenopyrite or pyrite. Sphalerite formed rare angular particles up to 110 microns. Chalcopyrite occurred as free grains up to 105 by 225 microns in size. It also formed middling particles with pyrite. The chalcopyrite exhibited exsolution lamellae to bornite.

Rutile and sphene exist as secondary minerals after ilmenite and titanomagnetite. Rutile commonly displayed twinning and formed coarse angular to subhedral particles. Sphene typically occurred as aggregates of anhedral subrounded grains. Baryte occurred as angular to subhedral particles (Figure 67).

Gold Mineralization in Freegold - TR20 Ore

Eleven occurrences of native gold were observed in the Freegold TR20 sample. The gold commonly formed free angular to smooth, subrounded (Figure 70) to lenticular elongate particles ranging from 10-20 microns and 80 - 180 microns (Figure 68). A subsurface dimension of 85-220 microns was observed in the largest free gold particle. The free gold typically averaged 35-40 microns in diameter (Figure 69). A coarse particle of 170-250 microns comprised an elongate narrow lenticular rim of gold around a massive core of gold associated with secondary iron-titanium oxides (Figure 73). The form may indicate a fracture filling mineralization mechanism. Gold particles may be attached to goethite-limonite particles or gangue minerals (Figure 72). A single free gold particle contained a minute inclusion of pyrrhotite. The typical composition of the gold particles is shown in Figure 71.

5.0 METALLURGICAL TEST WORK

5.1 Procedures

5.1.1 Sample Preparation

The samples were received as coarse bulk ore material from surface trenches of the Antoniuk (Mt. Freegold), Casino and Revenue deposits. The approximate weights of the sample are as follows:

Antoniuk (Mt. Freegold)		
	- TR20	1590 kg (3500 lb.)
	- TR18	1590 kg (3500 lb.)
	- TR24	1590 kg (3500 lb.)
	- TR21	1136 kg (2500 lb.)
Casino		1136 kg (2500 lb.)
Revenue		450 kg (990 lb.)

The bulk samples from each deposit (zone) were coned four times. A 50 kg representative sample was randomly split from the coned bulk sample for each deposit. The remainder of the coned ore sample was used for the heap leach test work.

Each zone sample was then screened on a six mesh screen and the oversize fraction was cone crushed until the samples were 100% minus six mesh in size. The samples were carefully blended and riffled into 500 gram lots to be used for analytical and bench scale metallurgical tests (agitated leach).

Duplicate 500 gram samples were cut from each head sample, half of which was pulverized to 100% - 150 mesh and the other half saved as reject. The -150 mesh material was submitted for duplicate fire assays, with DCP finish.

5.1.2 Grinding

The 500 gram samples were ground using a 10.3 kg ball charge in a 20 x 20 cm ball mill. Milling time was 15 minutes at 64 RPM. The ore was ground at 65% solids. It produced a feed of 85.6% passing 200 mesh for leach tests.

5.1.3 Agitated Leach Cyanidation

The ground slurry from the ball mill was washed into a 2 litre reaction kettle. The ore slurry was diluted to 50% solids and agitated using a Caframo stirrer with a three blade impellor. Lime was added to give the desired pH (10.5, 11.5). The sodium cyanide was added directly to the leach slurry to give a sodium cyanide concentration of 1.00 gram per litre. The pH was maintained at the desired level by adding lime prior to cyanide additions. The free cyanide concentrations were monitored throughout the tests by sampling settled solutions and titrating using a standard $AgNO_3$ solution. Titration solutions were combined and retained for gold analysis. Values are included in the overall mass balance calculations and are reported on the leach summary sheets (Appendix III).

At the end of the 24 hour leaching period, each slurry was vacuum filtered on a 12.5 cm diameter Buchner funnel. The cake was displacement washed with twice the dry solids weight (5-200 ml portions) of wash water. The volumes of filtrate, wash and titration solutions were taken and each solution was submitted for gold analysis. Cyanidation residues were dried at 110 °C and passed through a 0.417 mm (-35 mesh) sieve. Samples were blended by riffling and then split in half. Half of the sample was used as an analyical sample and half retained as reject for reference.

A duplicate fire assay analysis was carried out on each residue.

5.1.4 Heap Leach Cyanidation

The following procedure was used for the small pilot scale column heap leach cyanidation test work on four of the six bulk ore sample. These samples included:

1. Antoniuk (Mt. Freegold) - TR20
2. Antoniuk (Mt. Freegold) - TR24
3. Casino
4. Revenue

After the original bulk sample was coned, a 50 kg sample was collected for analytical and bench scale tests. The remaining ore sample was again coned and packed into the large scale columns. The columns, comprised of three stacked 45 gallon drums, were filled with ore. The sample weights were as follows:

1.	Antoniuk (Mt. Freegold) - TR20	833.5 kg
2.	Antoniuk (Mt. Freegold) - TR24	980 kg
3.	Casino	890 kg
4.	Revenue	391 kg

Additional sample was retained for the Antoniuk and Casino deposits while all of the Revenue sample was used for the heap leach cyanidation test. The columns were washed with 200-244 litres of water (see summary - Appendix IV) and lime only was added during the first twelve hours. The pH values were monitored and maintained between 9.9 and 10.6. Sodium cyanide was added after the initial twelve hour pre-leach stage. The sodium cyanide levels were started at 1.00 grams per litre, dropped to 0.50 grams per litre and raised to 1.00 grams per litre for 1 day prior to the end of the leach. The Revenue sample was treated with an initial cyanide level of 0.50 grams per litre. Cyanide and lime levels were monitored by using standard $AgNO_3$ titrations for sodium cyanide and oxalic acid titrations for lime. Reagents were added to maintain the desired levels and the consumptions were monitored. Water was added daily to compensate for evaporation and to maintain solution volume.

Solutions and gold loaded activated carbon samples were taken periodically and analysed for gold. From these results, a gold extraction curve was established. The leaching cycles were maintained for the following durations:

1.	Antoniuk TR20	15 Days
2.	Antoniuk TR24	44 Days
3.	Casino	27 Days
4.	Revenue	31 Days

After the solution flow to the columns was stopped, the columns were allowed to drain and were washed with 160-210 litres of fresh water (see summary - Appendix IV) for one Day. The barren solution was filtered and analysed for gold. The fines from the barren solution filter were weighed and combined with the residue.

The column was emptied and the ore sample residue was spread out on a plastic sheet to dry. Fifty kg samples of the residue was collected and washed through 2 inch, 1 inch and 1/2 inch screens to obtain four size fractions (see Appendix IV). The screen fractions were dried and weighed. The weights were recalculated to the original total sample weights (Appendix IV) and samples of the screen fractions were riffled and ground to minus 150 mesh for analysis.

Based on the assay results from the TR24 heap leach residue screen fractions (+1/2 inch, -1/2 inch), rewash procedures were undertaken to investigate the relationship between gold content, fine grained particulates, particle size (+ 1/2 inch or -1/2 inch) and gold extractions. The rewash procedures are described below.

+1/2" Fraction

The + 1/2" fraction was coned 3 times and then approximately 5 kg of sample was taken from opposite quarters. This 5 kg sample was repulped in 5 litres of pH^+ 10.5 water. The slurry was stirred for a few minutes and then filtered. The filtrate was sampled and analyzed for Au. The filter cake was dried, weighed and then crushed to -6 mesh. This 6 mesh material was then riffled into a 500 g representative sample and prepared to pass through a 150 mesh screen (100% -150). It was then split in half, and retention of reject.

-1/2" Fraction

This fraction was coned 4 times and one quarter was taken out to be further reduced. The quarter was then screened on a 6 mesh screen and the +6 mesh material was crushed until all the material passed through the screen. The sample was then coned three times and opposite quarters were combined to form the sample. Three 1000 gram samples were riffled out and the remainder retained.

Each of the three samples was repulped with 1000 mls of pH^+ 10.5 water. The slurry was stirred and then filtered. After all the solution was filtered, another 200 mls of pH^+ 10.5 water was added to wash the cake.

After filtering was completed, a sample of the filtrate was analyzed for gold. Each filter cake was dried, weighed and pulverized to 100% -150 mesh. One quarter of each sample was then riffled out and submitted for gold analysis.

5.2 Results

5.2.1 Agitated Leach Tests

The results of the agitated leach tests carried out on the Antoniuk (TR20, TR24), Casino and Revenue deposits are given in Tables 2 and 3. The summary leach sheets are included in Appendix III.

Table 2

CYANIDATION OF ARCHER - CATHRO ORES

AGITATED LEACH TESTS SUMMARY

Test (#)	Feed	Grind (%-200m)	pH	Consumpt (kg/t) NaCN	Consumpt (kg/t) Lime	Au Residue (g/t)	Au Calc Hd (g/t)	Au Extr'n (%)	Au Mass Bal. (%)
1	TR20N	85.6	10.5	2.36	0.81	tr	1.50	99.00	312.5
2	TR20N	85.6	11.5	1.42	2.09	tr	2.50	99.90	208.3
3	REVENUE	NA	10.5	1.67	2.9	0.33	0.57	44.9	49.8
4	REVENUE	NA	11.5	1.24	19.60	0.16	0.70	77.1	60.9
5	CASINO	NA	10.5	1.35	3.11	0.32	1.10	71.7	100
6	CASINO	NA	11.5	0.18	8.57	0.30	1.20	75.3	109.1
7	TR24N	NA	10.5	1.59	7.32	0.16	1.67	90.5	125.6
8	TR24N	NA	11.5	0.28	12.07	0.26	1.77	86.2	133.1

Table 3

IMPURITY LEVELS IN FINAL LEACH SOLUTIONS

Test No.	Sample	pH+	Cu	mgpl Fe	Zn
1	TR20N Zone 2	10.5	3.03	-	1.94
2	"	11.5	2.93	-	1.59
3	REVENUE	10.5	301	-	1.96
4	"	11.5	167	-	0.62
5	CASINO	10.5	55.3	-	1.48
6	"	11.5	33.8		1.38
7	TR24N Zone 3	10.5	2.59	-	-
8	"	11.5	2.12	-	-
HL-1	TR20N	10.5	16.0	4.24	3.12
HL-2	REVENUE	10.5	332.5	5.32	3.64
HL-3	CASINO	10.5	131	19.8	3.39
HL-4	TR24N Zone 3	10.5	1.74	-	-

5.2.2 Heap Leach Tests

The results of the column heap leach tests are summarized in Table 4. The summary leach sheets are included in Appendix IV.

Gold extraction data are given in Table 5. The screen fraction analysis and gold distribution within the heap leach residue are shown in Table 6. The Freegold TR24 residue rewash results are given in Table 7.

Table 4

CYANIDATION HEAP LEACH TESTING RESULTS

Test No.	Sample	pH+	Consumptions kg/t NaCN	Consumptions kg/t Lime	Au Res. g/t	Au Calc Hd g/t	Au Mass Bal. %	Au Extr'n %
1	TR20N Zone 2	9.9-10.5	0.79	0.15	0.03	0.69	97.2	95.6
2	REVENUE	9.9-10.5	2.18	0.95	0.20	0.38	33.0	47.1
3	CASINO	9.9-10.5	1.17	0.64	0.71	0.93	84.5	23.0
4	TR24N Zone 3	9.9-10.5	1.40	0.71	1.016	1.65	124.1	38.5

Table 5

Au EXTRACTION VS TIME FOR HEAP LEACH TESTS

TR20N HL-1

Time Days	Cumulative mg	Au Extr'n %
1	317.64	55.6
2	392.94	68.8
7	439.5	76.9
13	545.3	95.4
15	546.5	95.6

REVENUE HL-2

Time Days	Cumulative mg	Au Extr'n %
1	12.13	8.2
4	25.04	16.9
10	35.91	24.3
18	48.07	30.5
25	60.06	40.6
31	69.72	47.1

CASINO HL-3

Time Days	Cumulative mg	Au Extr'n %
1	43.4	5.3
3	59.06	7.2
6	65.61	8.0
14	134.32	16.4
21	162.44	19.9
27	187.9	23.0

TR24N HL-4

Time Days	Cumulative mg	Au Extr'n %
1	106.01	6.5
7	225.91	13.9
14	419.36	25.9
32	533.95	33.0
37	539.74	33.3
44	622.52	38.5

Table 6

HEAP LEACH RESIDUE
SCREEN FRACTION ANALYSIS
GRADE CALCULATION

TR20N ZONE 2 HL-1

Screen Size	Wt (kg)	Au (g/t)	Au (mg)	% Distribution	
				Wt	Au
+2"	299.5	0.03	8.99	35.9	35.9
-2 + 1"	172.7	0.03	5.18	20.7	20.7
-1" + 1/2"	91.6	0.03	2.75	11.0	11.0
- 1/2"	270.2	0.03	8.11	32.4	32.4
	833.5	(0.03)	25.03	100.0	100.0

REVENUE HL-2

Screen Size	Wt (kg)	Au (g/t)	Au (mg)	% Distribution	
				Wt	Au
+2"	79.5	0.32	25.44	20.6	32.6
-2 + 1"	102.0	0.29	29.58	26.4	37.8
-1" + 1/2"	44.0	0.05	2.20	11.4	2.8
- 1/2"	161.0	0.13	20.93	41.6	26.8
	386.5	(0.20)	78.15	100.0	100.0

CASINO HL - 3

Screen Size	Wt (kg)	Au (g/t)	Au (mg)	% Distribution	
				Wt	Au
+2"	655	0.75	498.75	75.4	79.2
-2 + 1"	108	0.56	60.48	12.1	9.6
-1" + 1/2"	21.5	0.52	11.18	2.4	1.8
- 1/2"	88	0.67	58.96	10.0	9.4
	882.5	(0.71)	629.37	100.0	100.0

TR24N ZONE 3 HL-4

Screen Size	Wt (kg)	Au (g/t)	Au (mg)	% Distribution	
				Wt	Au
+ 1/2"	353	0.48	169.44	36.0	20.2
- 1/2"	627	1.07	670.89	64.0	79.8
	980	(0.86)	840.33	100.0	100.0

Table 7

HEAP LEACH RESIDUE - REWASH RESULTS

+1/2 inch fraction:		Au (ppm)
	Residue	0.37 *
	Filtrate	0.026

* Average of 0.29, 0.45

-1/2 inch fraction:		
	Residue 1	1.23
	Residue 2	1.34 *
	Residue 3	1.59
	Filtrate 1	0.08
	Filtrate 2	0.09
	Filtrate 3	0.08

* Average of 1.48, 1.20

Screen Distribution of Residue

Mesh	Wt.(kg)	Au (gpt)	% Distribution	
			Wt.	Au
+1/2 inch	353	0.37	36.00	13.12
-1/2 inch	627	1.38	64.00	86.88
	980	1.016	100.00	100.00

6.0 CONCLUSIONS AND RECOMMENDATIONS

The dye penetration test work indicated excellent potential for heap leaching in the Antoniuk-Freegold TR20 and TR18 zones while the TR24, TR21, Casino and Revenue deposits represent poor material for heap leaching of coarse uncrushed feed sample. Strong fracture penetration was displayed in the TR20 zone with weak fracture penetration in the TR24, TR21, Revenue and Casino deposits. Matrix penetration of the TR18 zone was well developed and indicated excellent heap leach potential. Matrix penetration of the other zones was marginal to weakly developed adjacent to fracture surfaces. The heap leach potential of these zones will be limited by the presence of fracture controlled gold mineralization.

The mineralogical studies of the Revenue and Freegold TR20 deposits indicated the presence of coarse free gold locally associated with secondary iron oxides and hydroxides. No gold occurrences were observed in the vanned concentrate of the Casino deposit. The presence of coarse free gold associated with the oxidized material is commonly indicative of high heap leach potential. However, due to poor matrix penetration of the fluorescent dye, the presence of fracture controlled mineralization is required for good extraction from all but the Freegold TR18 zone.

The agitated leach tests indicated that the gold extractions varied markedly between the three deposits. Direct cyanidation of Freegold TR20 indicate that gold extractions greater than 99% can be achieved. The gold, which occurs as coarse free particles, along mineralized fractures, exhibited a minute increase in recovery with high pH of 11.5. Mass balances indicate problems with the coarse gold particles. The Casino and TR24 agitated leach tests exhibited recoveries of 71.7 - 75.3% and 90.5 - 86.2%, respectively. The higher pH of 11.5 resulted in similar gold extractions though strong increases in lime consumption and decreases in cyanide consumption were observed. The lower recoveries may indicate the presence of finely disseminated locked gold mineralization. Agitated leach tests of the Revenue deposit exhibit large differences of 44.9% and 77.1% gold extractions at pH 10.5 and 11.5 respectively. The results may be indicative of solution fouling with copper, absence of free cyanide, and the presence of finely disseminated gold mineralization. The low mass balances may again indicate problems with coarse gold particles.

The heap leach tests achieved high recoveries of 95.6% for the Freegold TR20 deposit over 15 days, while the Revenue, Casino and Freegold TR24 deposits had moderate to poor recoveries of 47.1, 23.0 and 38.5% respectively. The Freegold TR20 deposit, characterized by abundant coarse free gold and fracture controlled mineralization provided excellent results over a short leach duration. The Revenue deposit may be limited by cyanide complexing with solubilized copper during the leach test. Optimization of the cyanide content may increase the recoveries, although the limited permeability and the results of the agitated leach tests indicate that maximum potential recoveries will be less than 75%. The Casino deposit, with a low recovery of 23%, exhibited poor permeability and low fracture density. Finer feed sizing would appear to be necessary for improved recovery in both the Revenue and Casino deposits.

The heap leach tests for the TR24 ore sample indicated low recoveries of 38.5% to 50.0%. The range was defined after several rewash tests of the leach residue screen fractions. Assays of the +1/2 inch and -1/2 inch screen fractions indicated that the -1/2 inch fraction had a higher gold content (3-5 times higher). The implications of the assay results are unclear, but may be due to mineralogical complications, presence of unleached coarse gold particles, small scale ore body zoning and/or spatial relationships between primary mineralization and fracture or shear density in the trench zones.

It is recommended that the further work on the Antoniuk Freegold deposit include mineralogical investigations on screen fractions of residues and feed samples to determine the type of gold mineralization. Further dye penetration test work should be used to define permeability variations in the deposit which may limit the grain size of material used on the heap leach tests.

APPENDIX I

PERMEABILITY TEST PHOTOGRAPHS

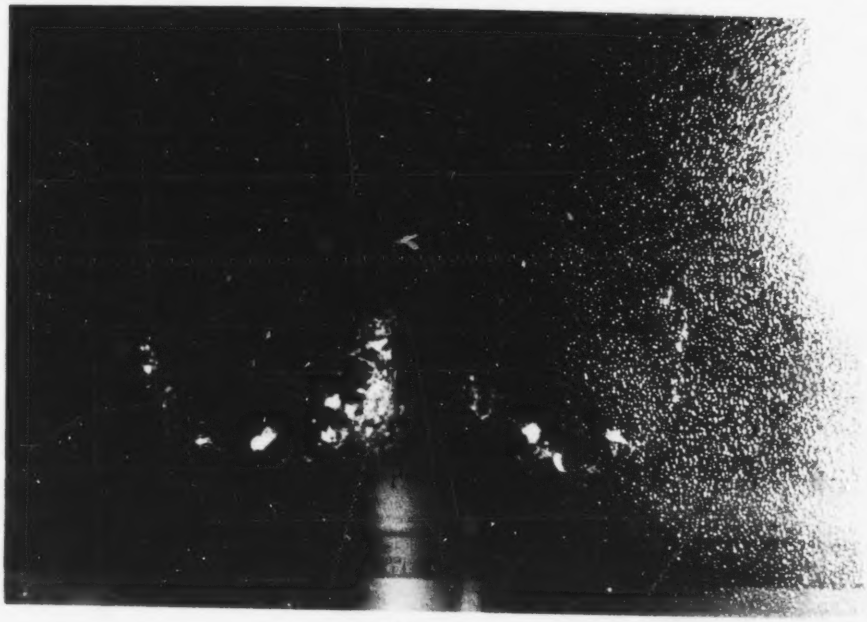

Figure 1
Sample - Revenue Deposit
Exposure - 1 Day

Figure 2
Sample - Revenue Deposit
Exposure - 3 Days

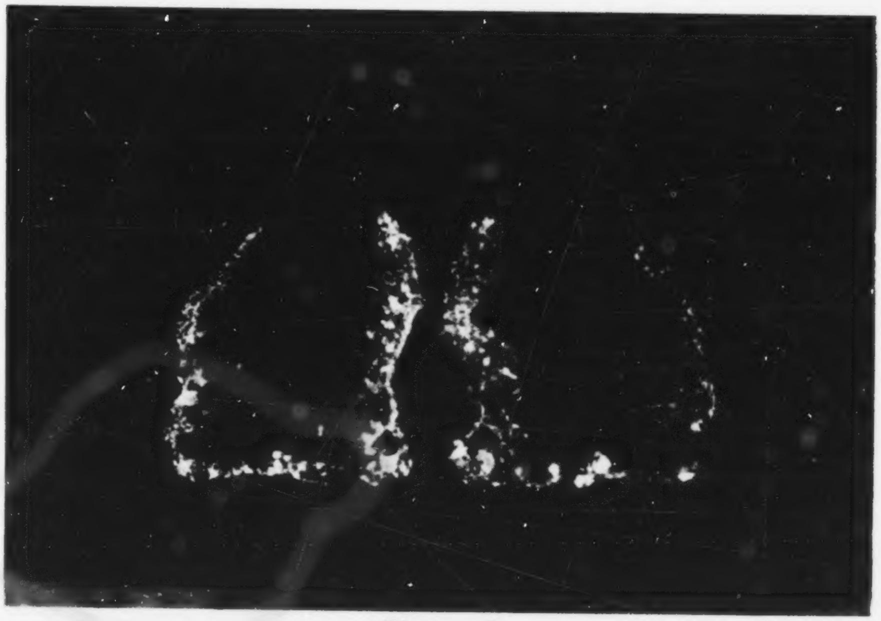

Figure 3
Sample - Revenue Deposit
Exposure - 7 Days

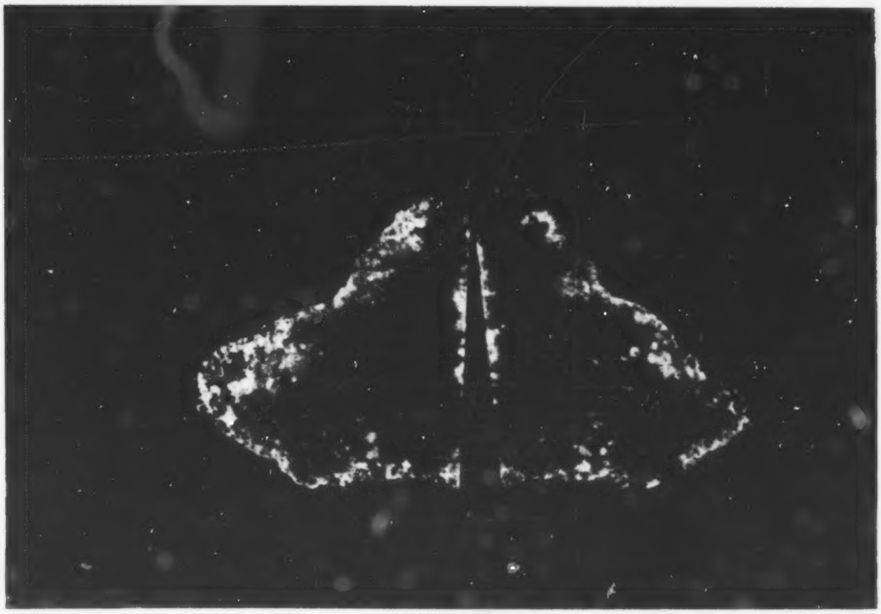

Figure 4
Sample - Revenue Deposit
Exposure - 15 Days

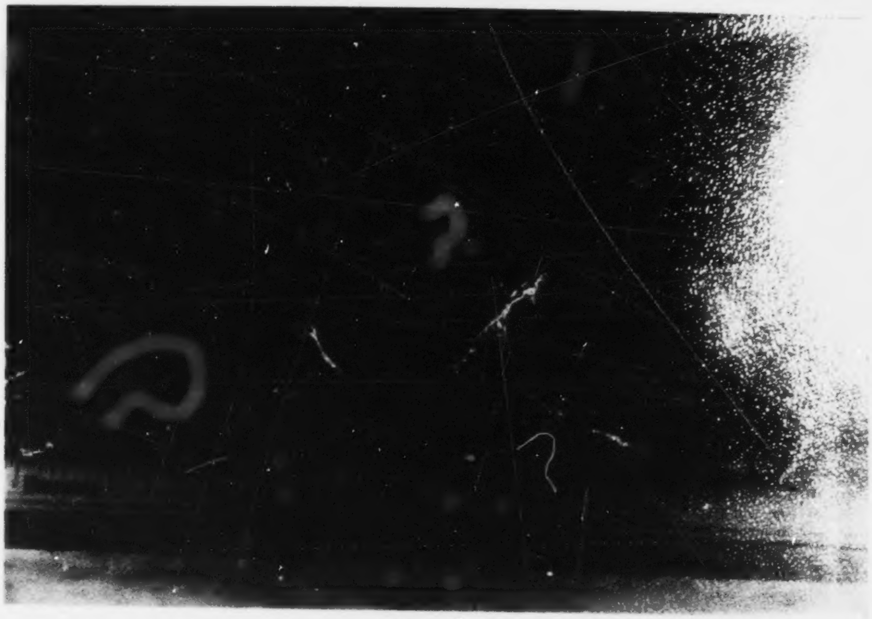

Figure 5
Sample - Casino Deposit
Exposure - 1 Day

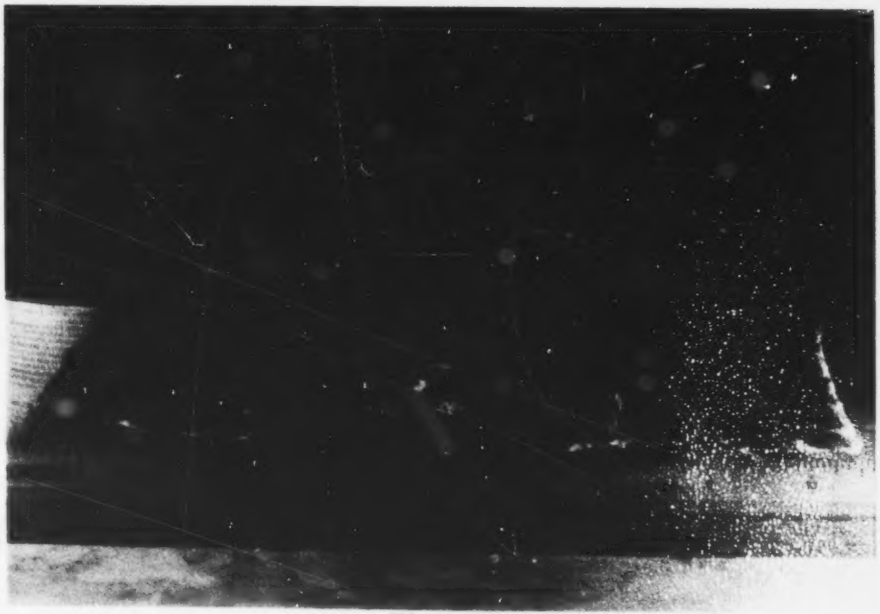

Figure 6
Sample - Casino Deposit
Exposure - 3 Days

Figure 7
Sample - Casino Deposit
Exposure - 7 Days

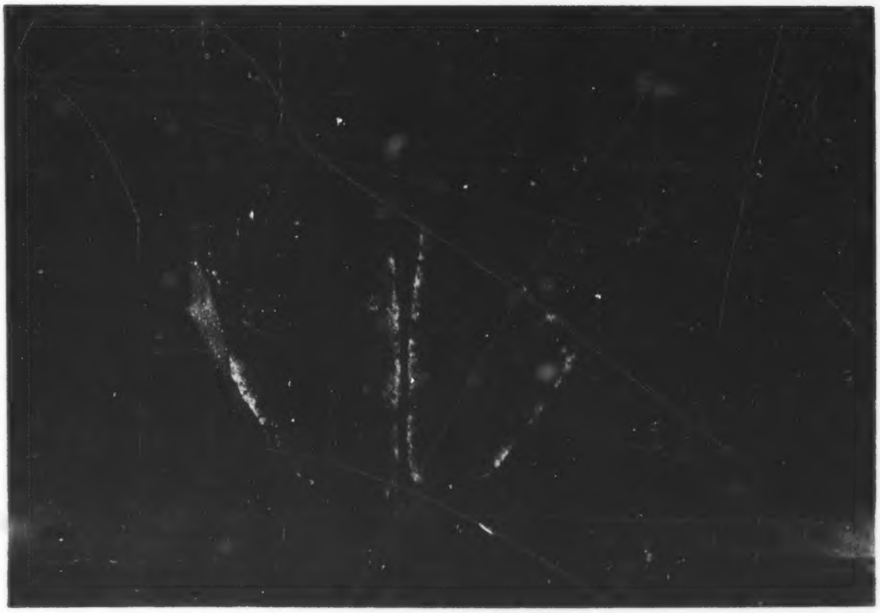

Figure 8
Sample - Casino Deposit
Exposure - 15 Days

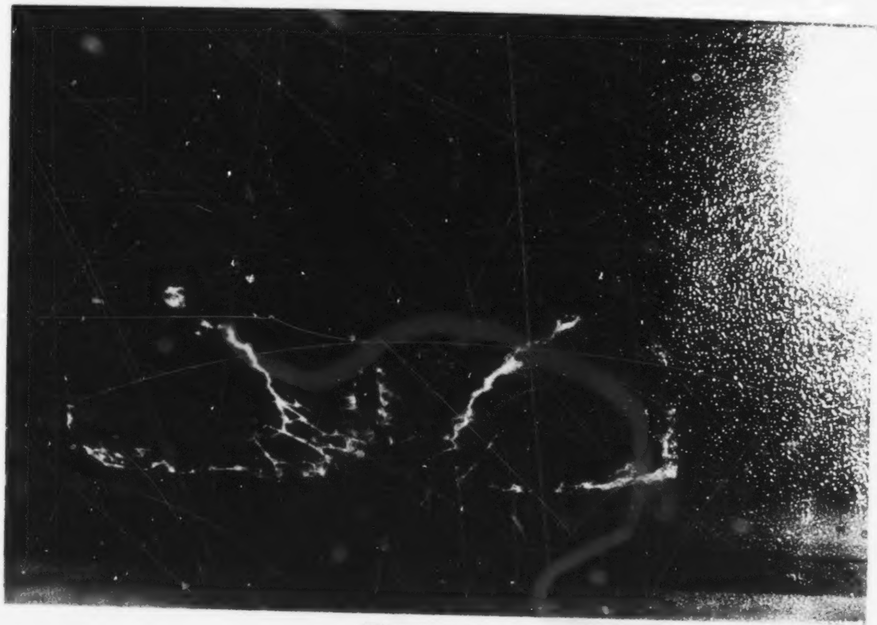

Figure 9
Sample - Freegold TR20 zone
Exposure - 1 Day

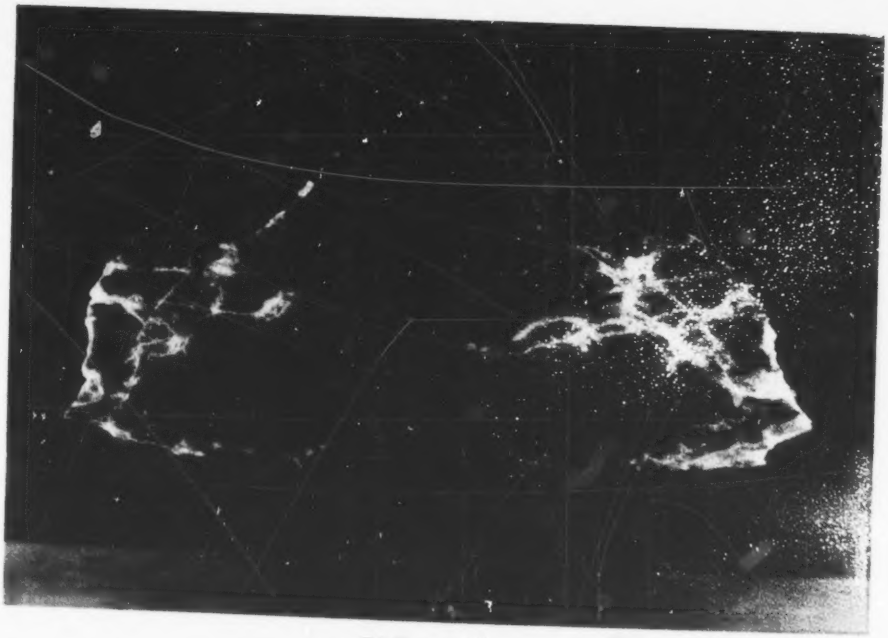

Figure 10
Sample - Freegold TR20 zone
Exposure - 3 Days

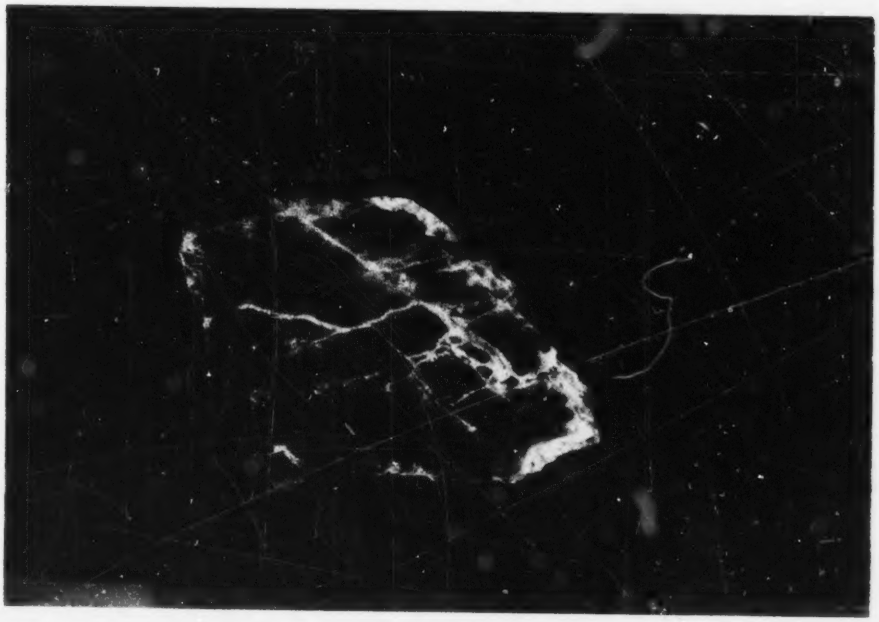

Figure 11
Sample - Freegold TR20 zone
Exposure - 7 Days

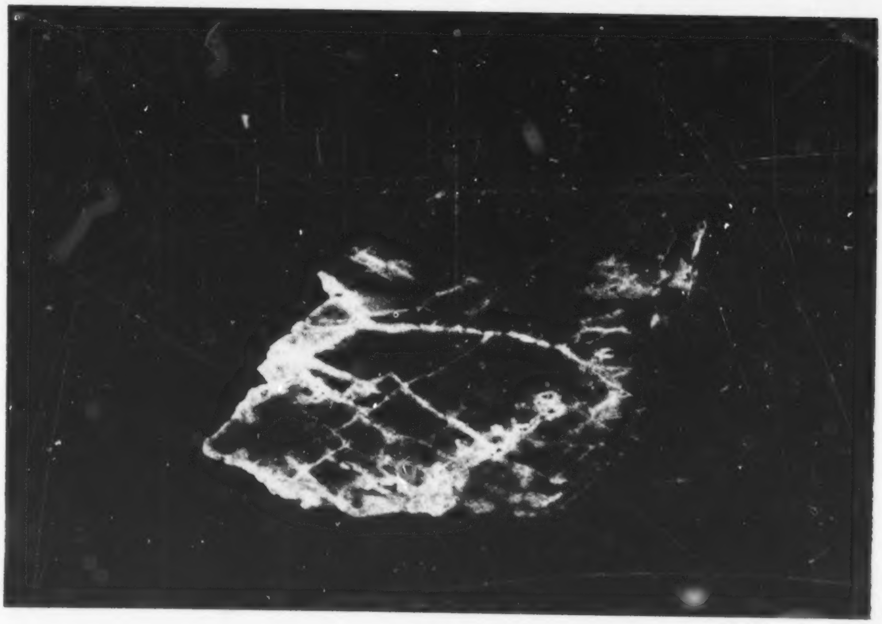

Figure 12
Sample - Freegold TR20 zone
Exposure - 15 Days

Figure 13
Sample - Freegold TR21 zone
Exposure - 1 Day

Figure 14
Sample - Freegold TR21 zone
Exposure - 4 Days

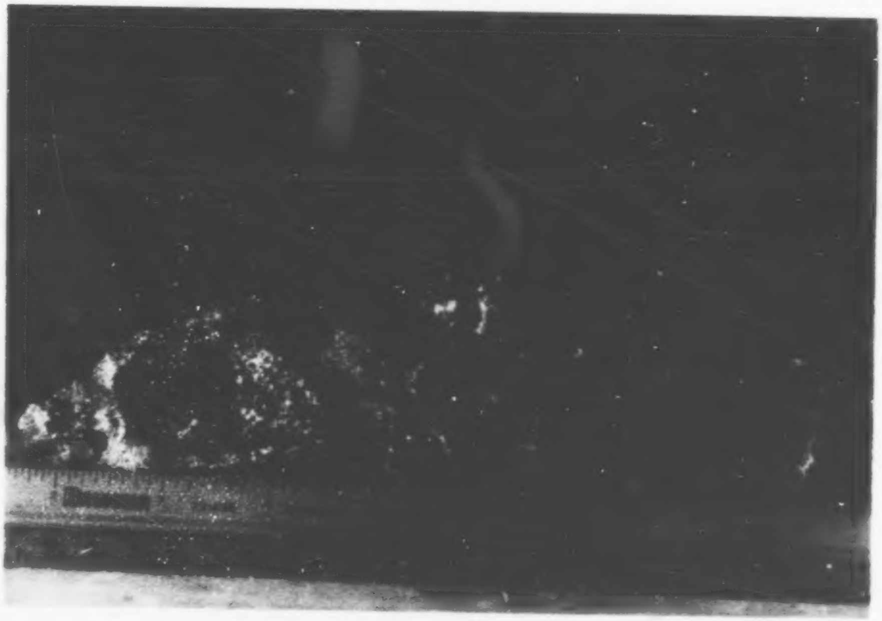

Figure 15
Sample - Freegold TR21 zone
Exposure - 8 Days

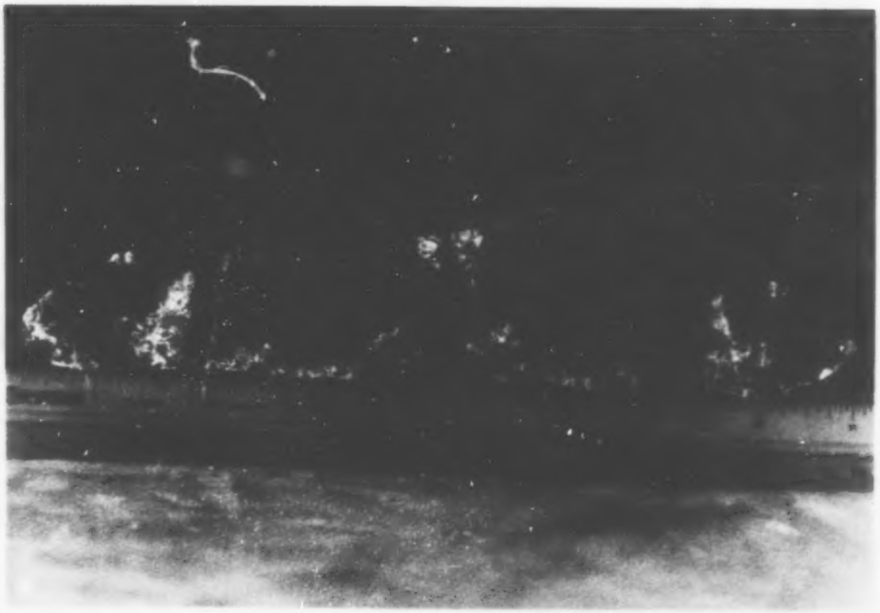

Figure 16
Sample - Freegold TR21 zone
Exposure - 11 Days

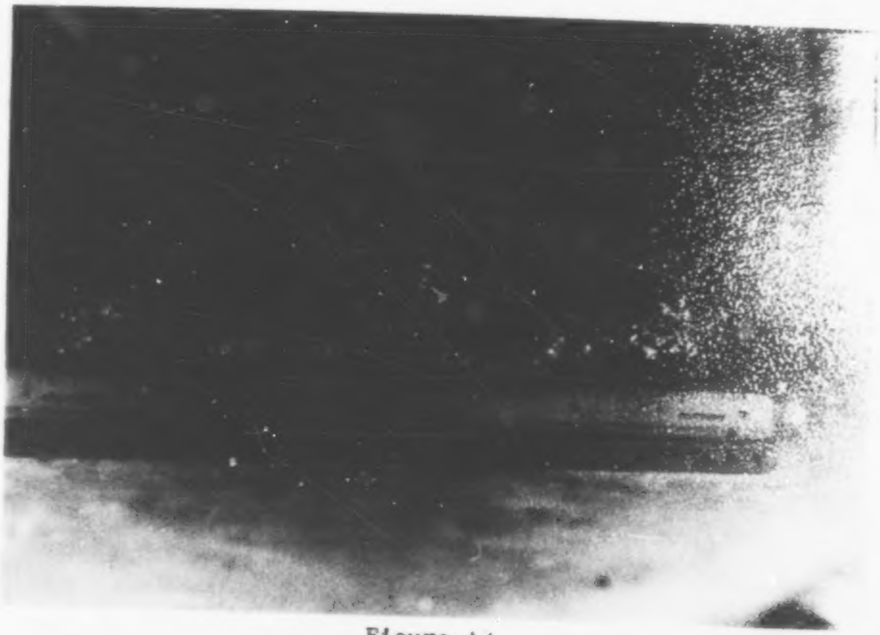

Figure 17
Sample - Freegold TR18 zone
Exposure - 1 Day

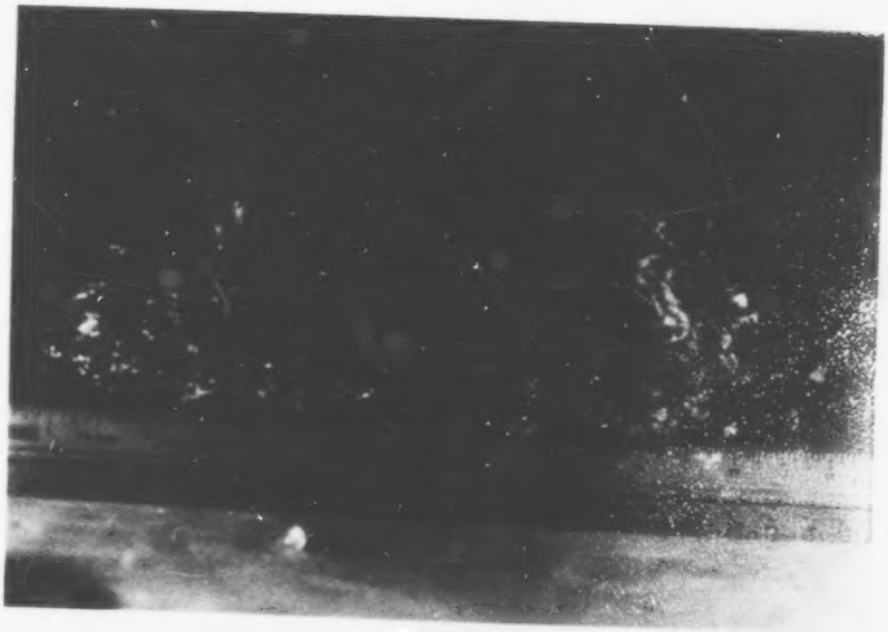

Figure 18
Sample - Freegold TR18 zone
Exposure - 4 Days

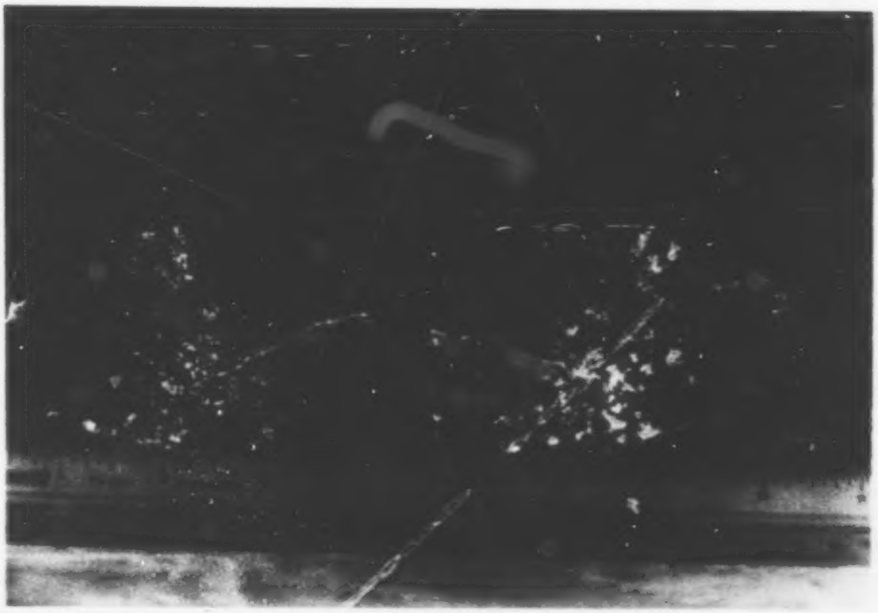

Figure 19
Sample - Freegold TR18 zone
Exposure - 8 Days

Figure 20
Sample - Freegold TR18 zone
Exposure - 11 Days

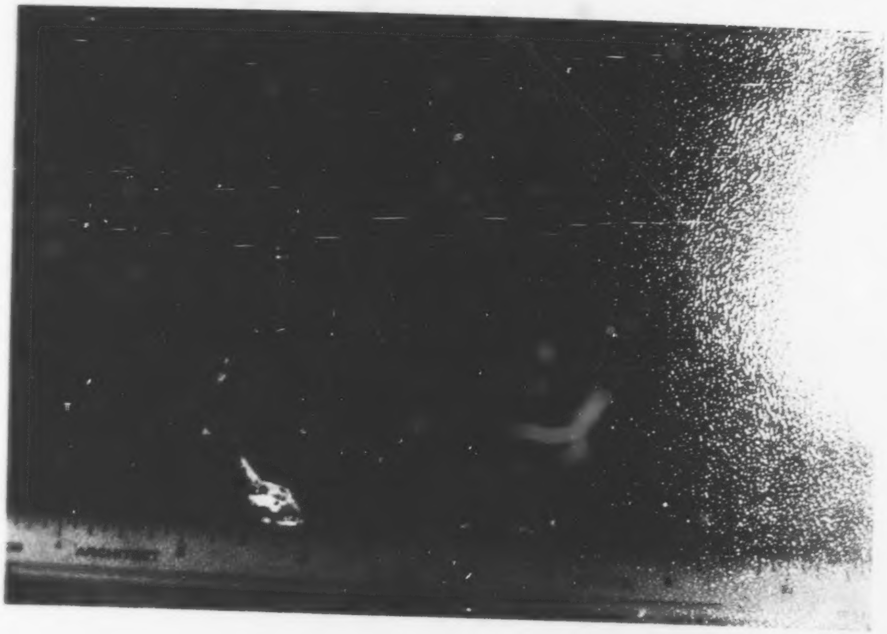

Figure 21
Sample - Freegold TR24 zone
Exposure - 1 Day

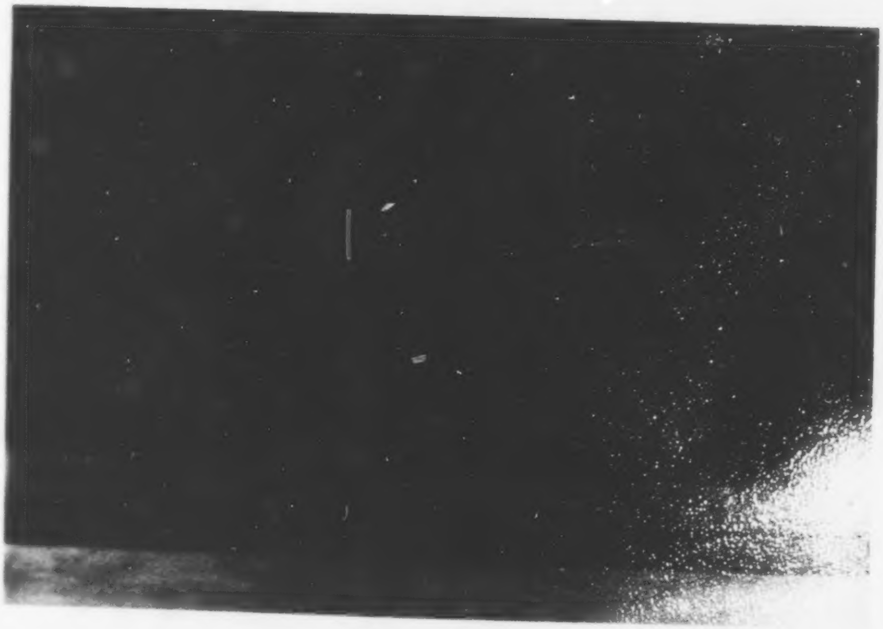

Figure 22
Sample - Freegold TR24 zone
Exposure - 3 Days

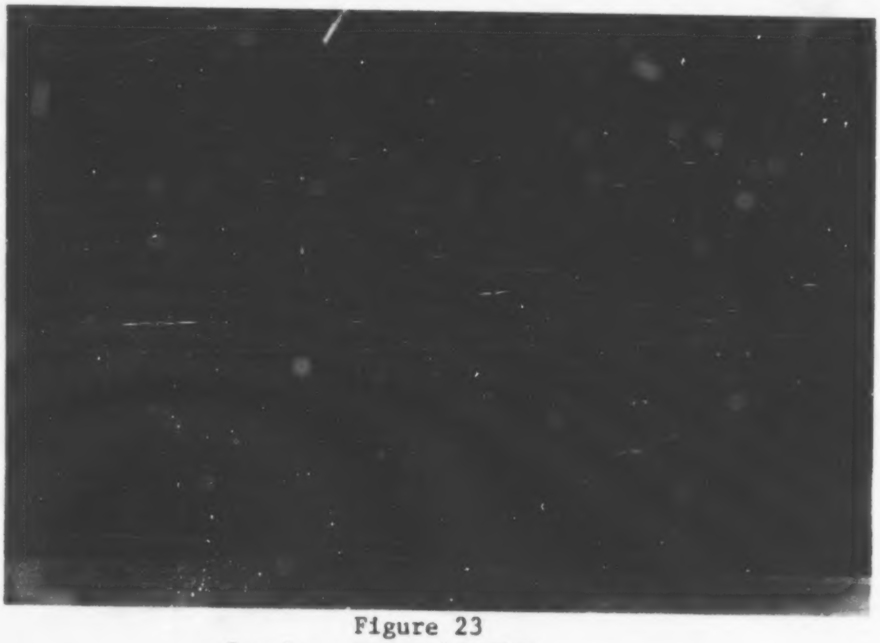

Figure 23
Sample - Freegold TR24 zone
Exposure - 7 Days

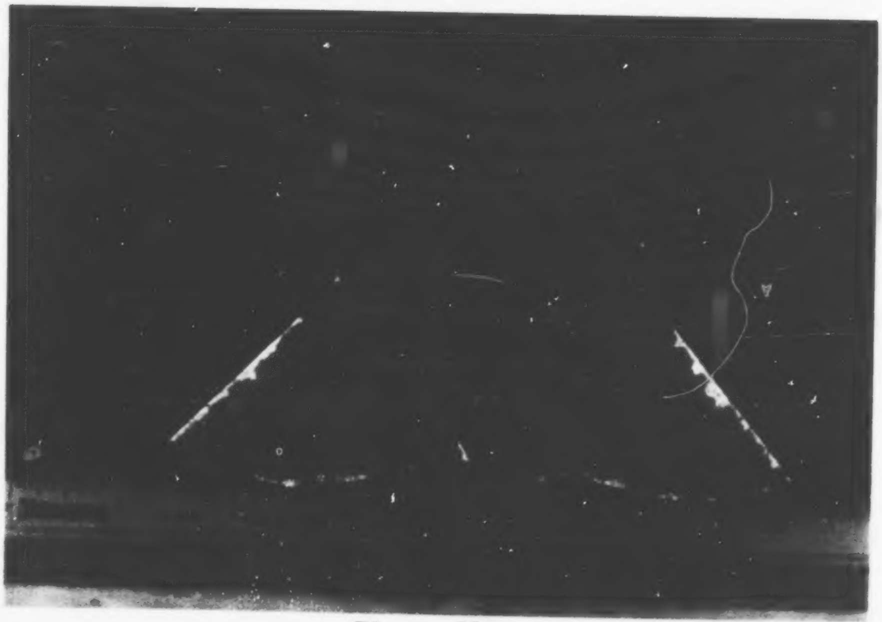

Figure 24
Sample - Freegold TR24 zone
Exposure - 15 Days

APPENDIX II

PHOTOMICROGRAPHS AND EDXA SPECTRAL PLOTS

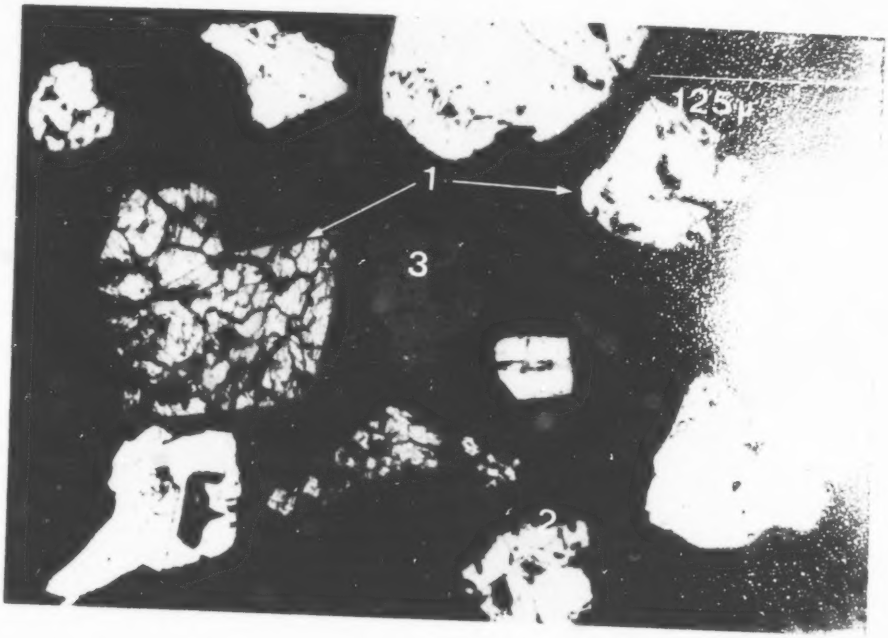

Figure 25
Subangular to subhedral particles of pyrite (1) exhibit moderate to intense fracturing and crustification to partial replacement by secondary iron oxides. Pyrrhotite (2) and magnetite (3) anhedra are also preseent.

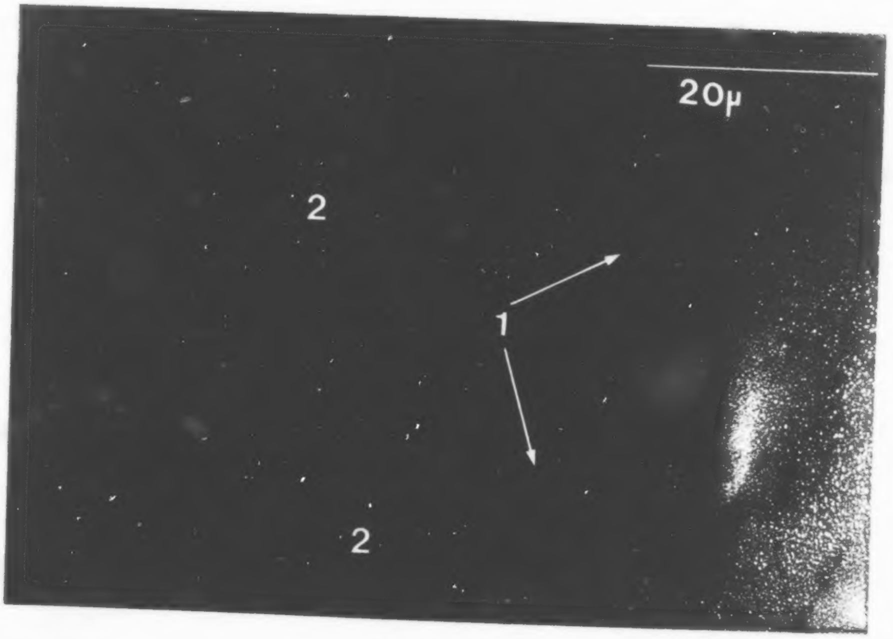

Figure 26
Numerous minor inclusions of galena (1) within pyrite (2). Inclusions of pyrrhotite and copper minerals are present and are typically less than 2-3 microns.

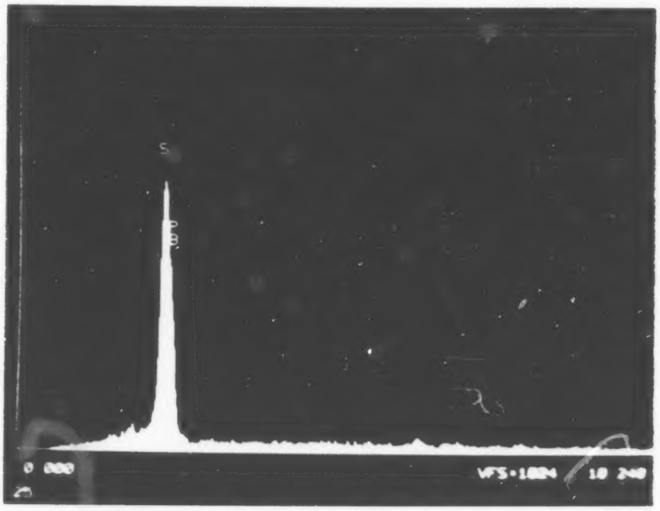

Figure 27
SEM-EDAX spectra of galena inclusion within pyrite.

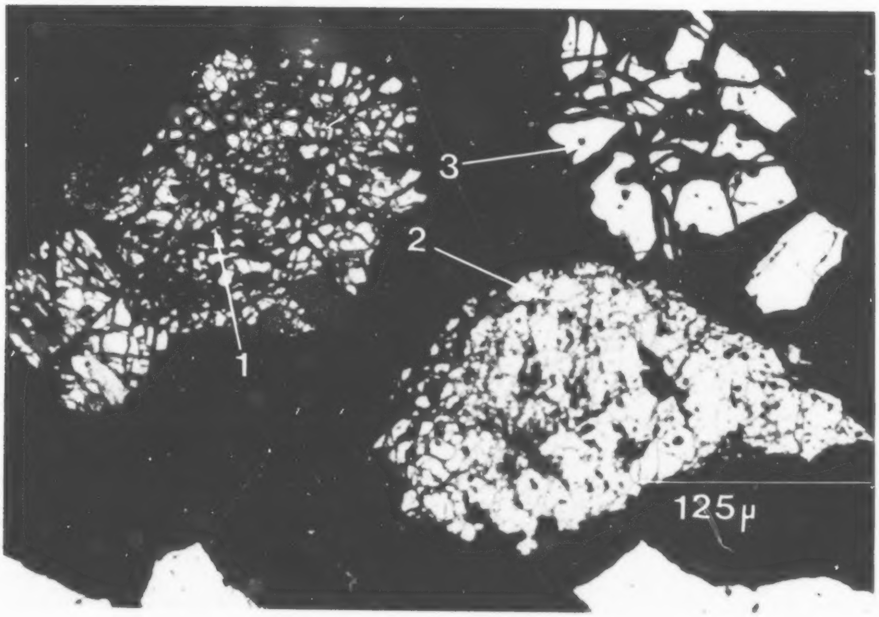

Figure 28
Pyrite (1), marcasite (2) and arsenopyrite (3) display progessive fracture controlled replacement by oxides, primarily goethite, hematite and minor hydroxides.

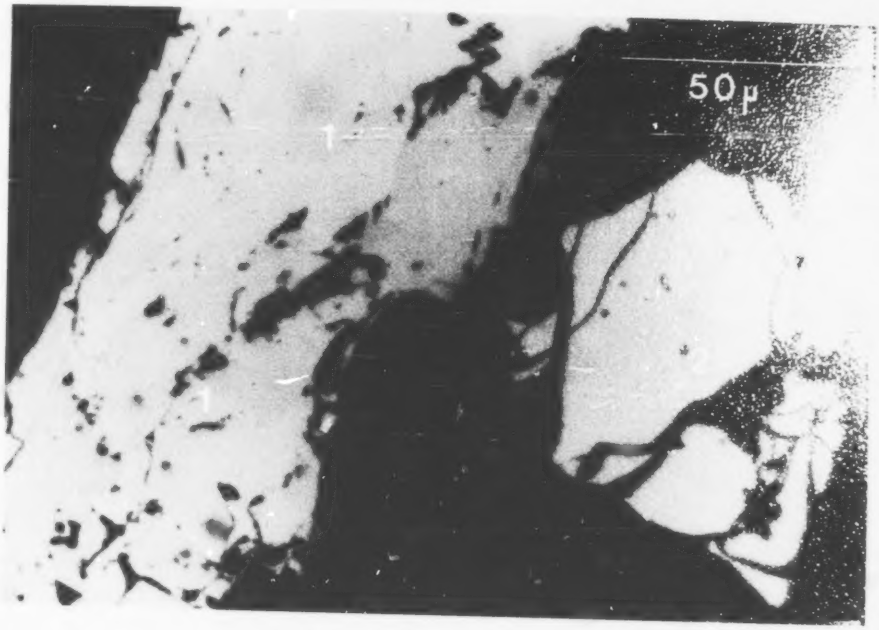

Figure 29
Fractured particle of marcasite (1) with well developed pleochroism and arsenopyrite (2) with fracture controlled oxide replacements.

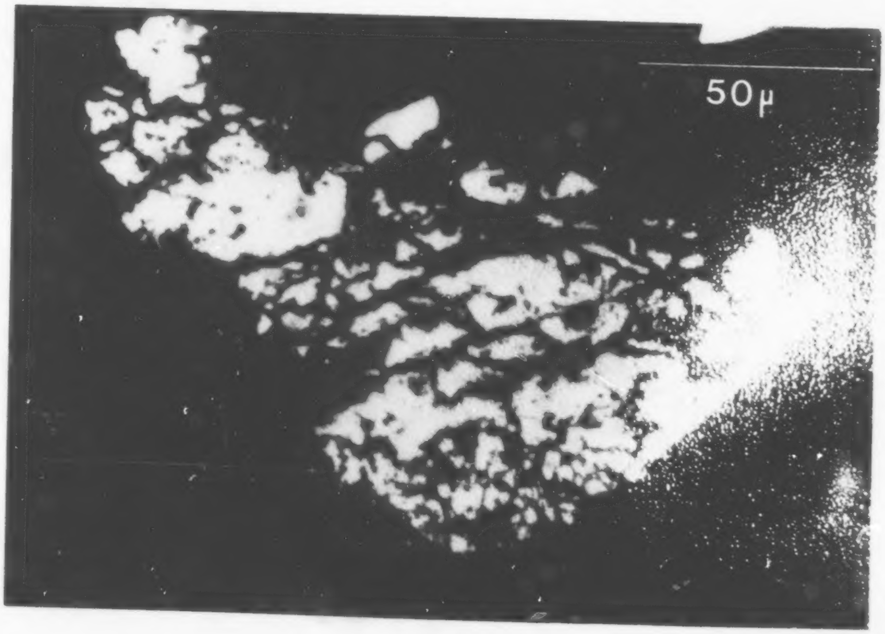

Figure 30
Strongly developed replacement of marcasite exhibits granular texture of original fine grained sulfides.

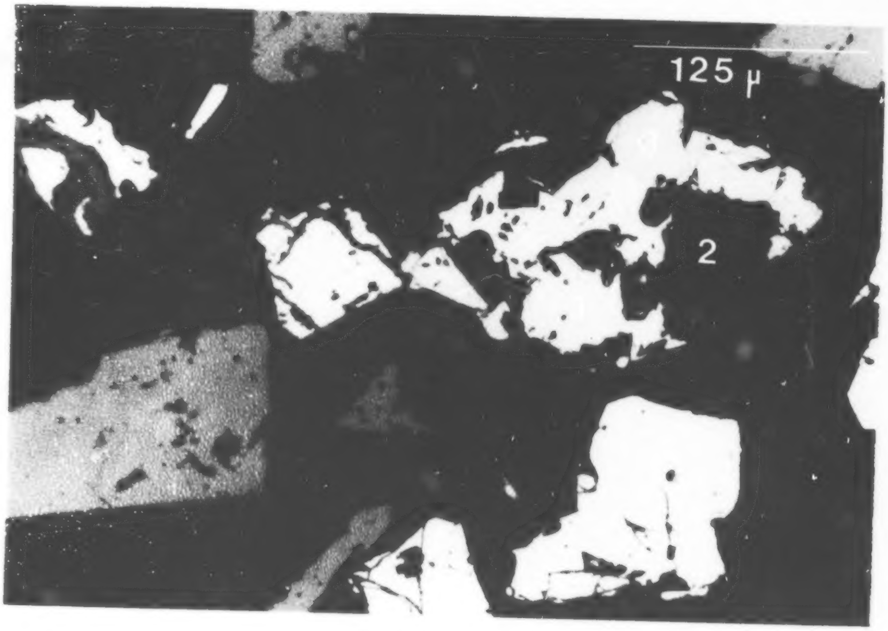

Figure 31
Angular subhedral chalcopyrite (1) attached to gangue (2) and subrounded anhedra of pyrrhotite (3). Chalcopyrite probably formed interstitial to pyrrhotite grains.

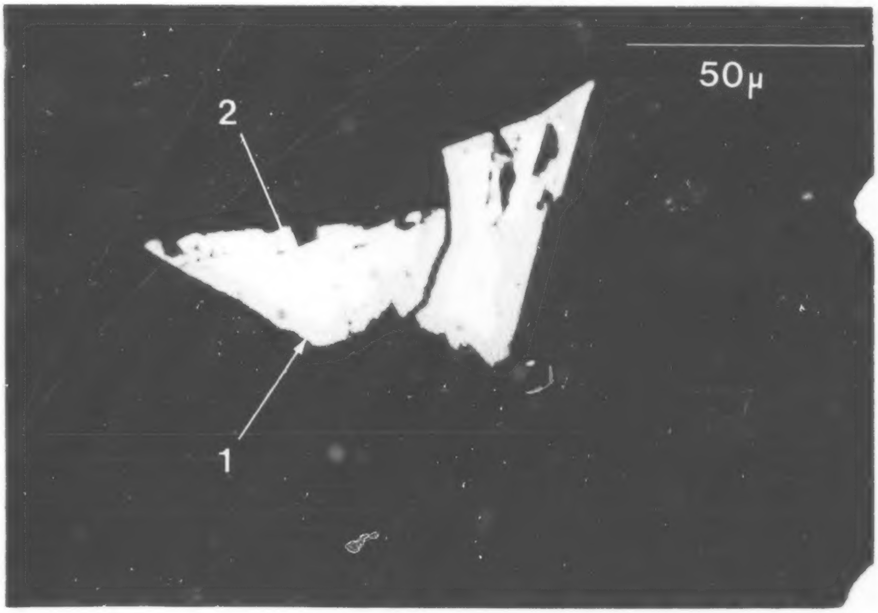

Figure 32
Pyrrhotite (1) formed porous, highly fractured (volume shrinkage) crusts replacing pyrite anhedra (2).

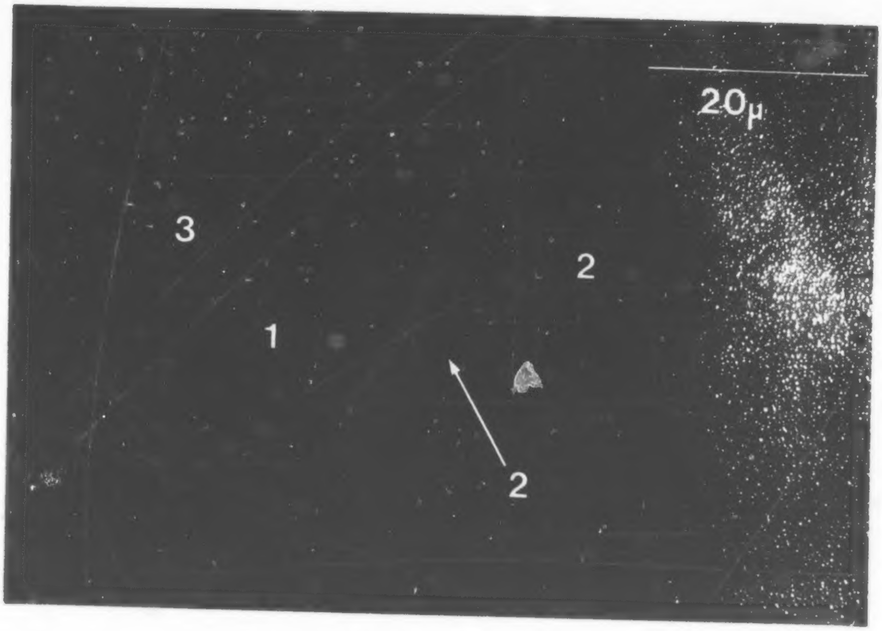

Figure 33
Elongate inclusion of chalcopyrite (1) and pyrrhotite exhibiting lamellae of pyrrhotite (2) within pyrite (3).

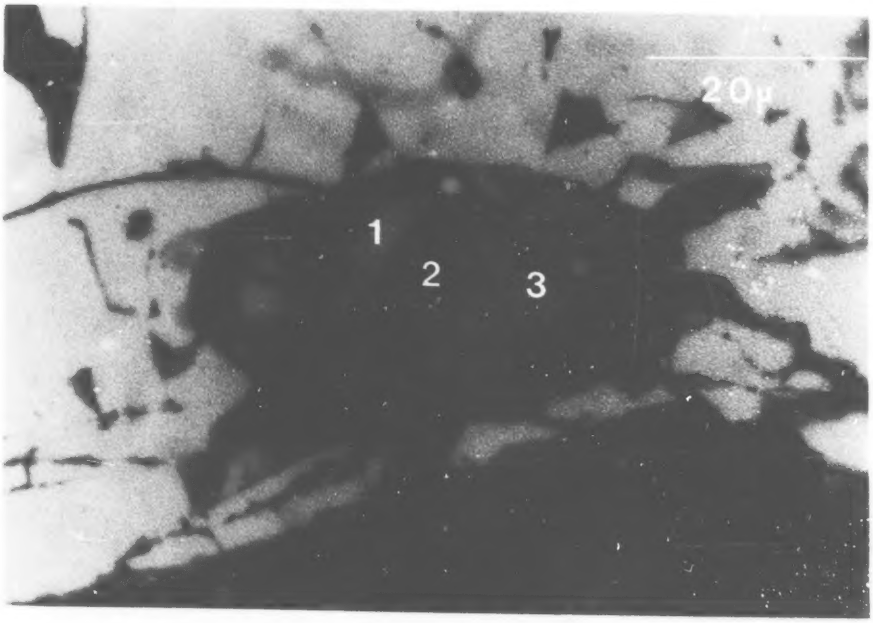

Figure 34
Smooth oblate inclusion of chalcopyrite (1), goethite (2) and rutile (3) within pyrite.

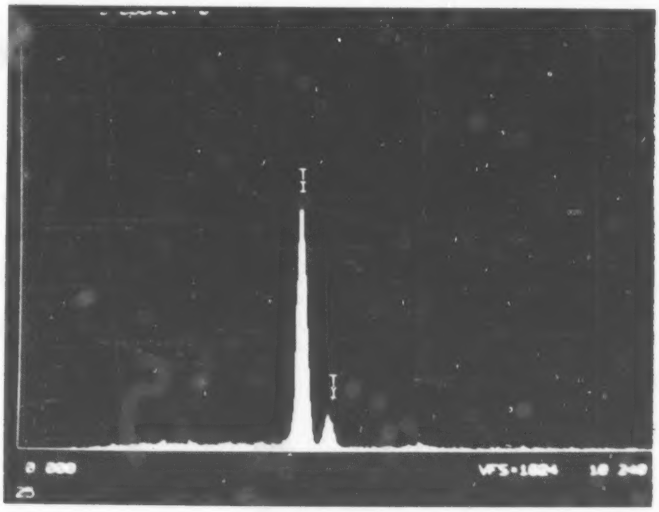

Figure 35
SEM-EDAX spectra of rutile.

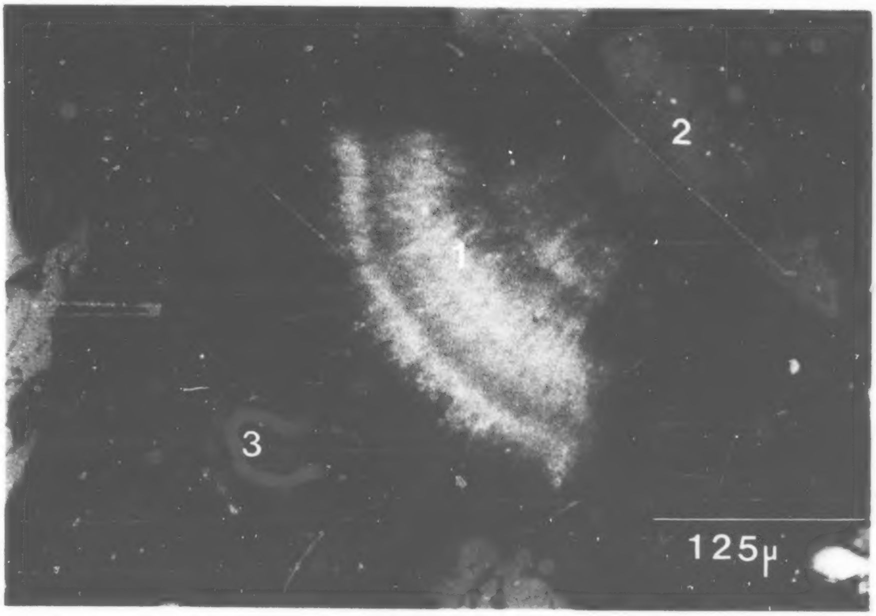

Figure 36
Angular fragment of malachite (1) exhibits radiating blades; also well developed collomorphic layering. Fragments of pyrite (2) and magnetite (3) are present.

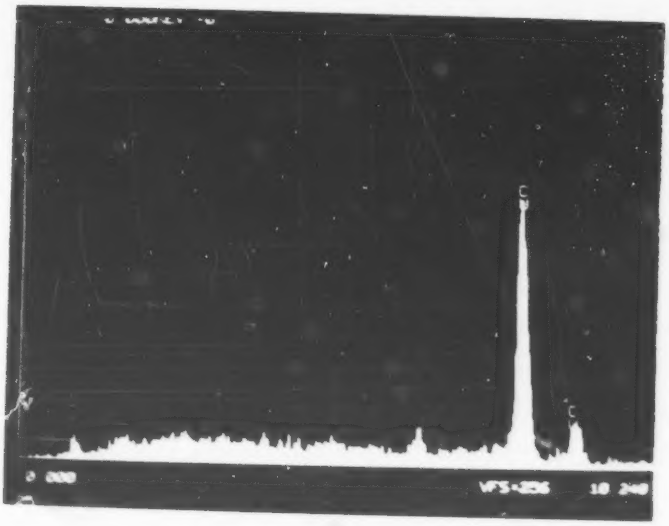

Figure 37
SEM - EDAX spectra of malachite.

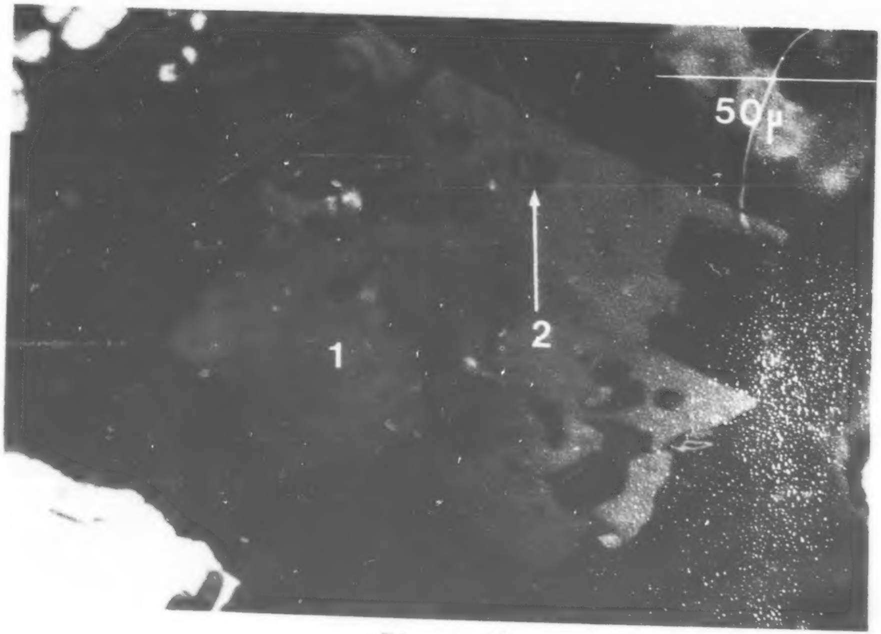

Figure 38
Angular free particle of sphalerite (1) with minor inclusion of bornite (2).

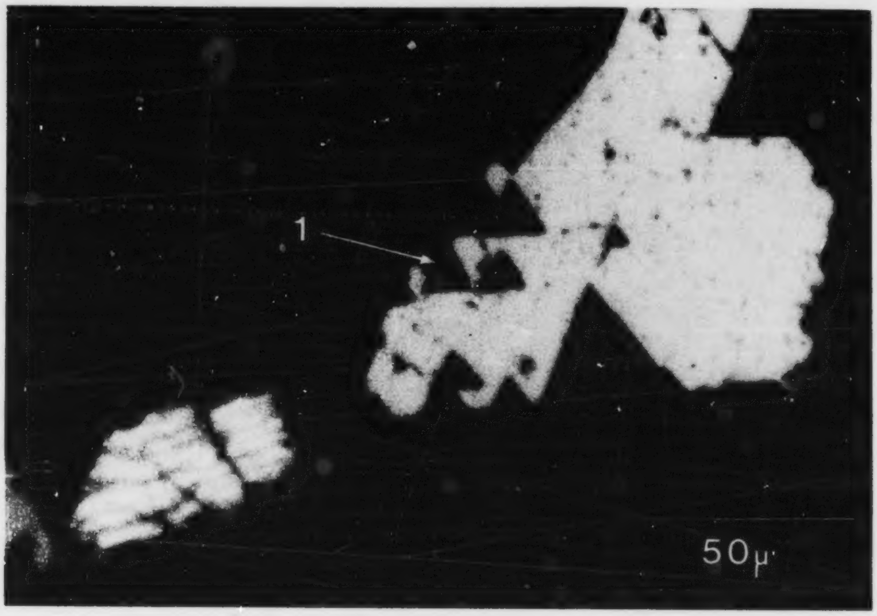

Figure 39
Angular subhedral particle of galena (1) exhibits triangular pits due to plucking at cleavage intersections.

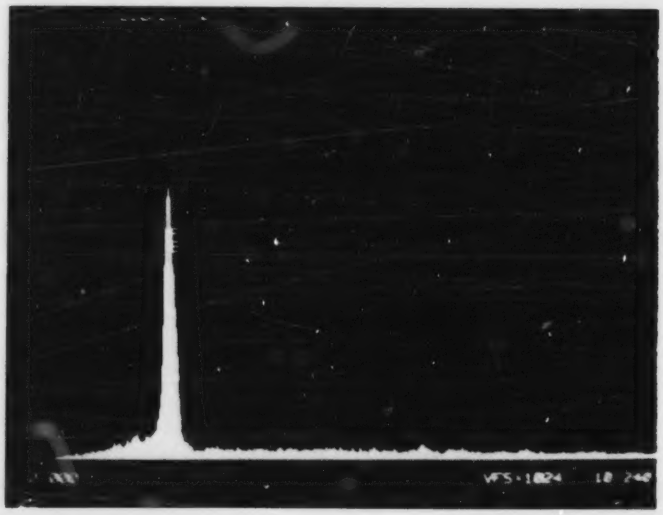

Figure 40
SEM-EDAX spectra of galena.

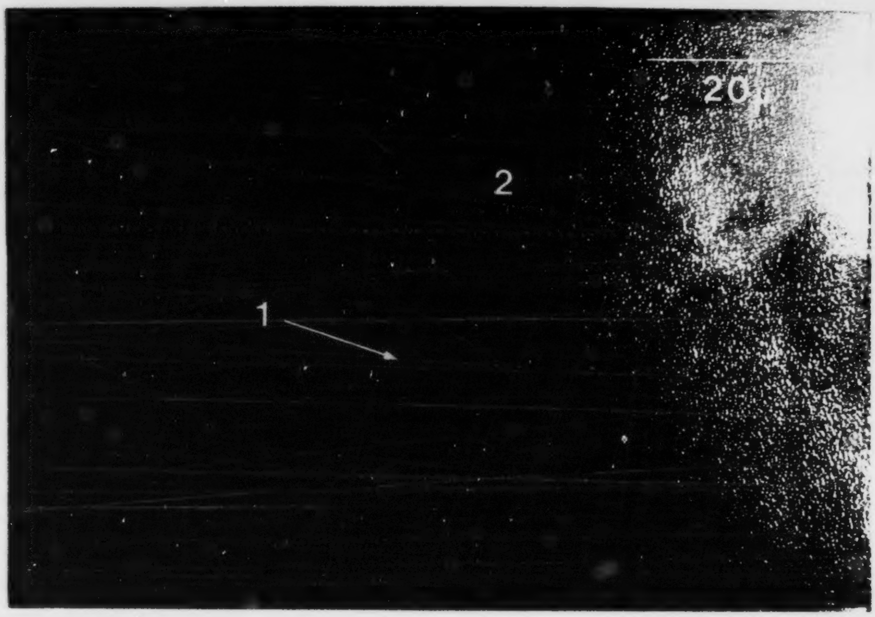

Figure 41
Minute 4 x 7 micron inclusion of tellurobismuthite (1) within pyrite (2).

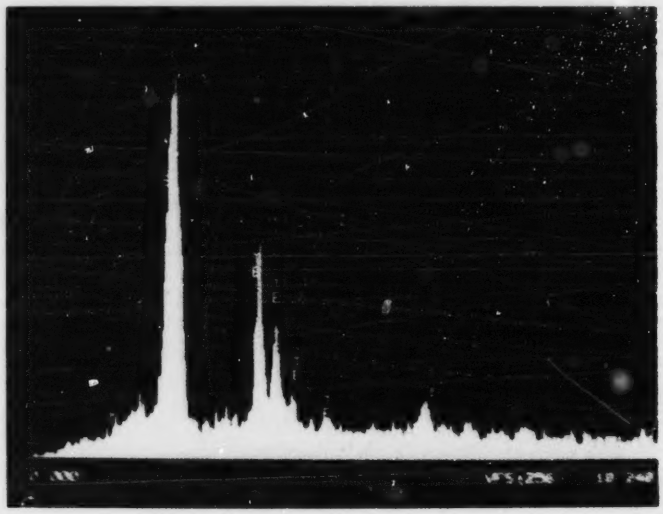

Figure 42
SEM-EDAX spectra of tellurobismuthite inclusion.

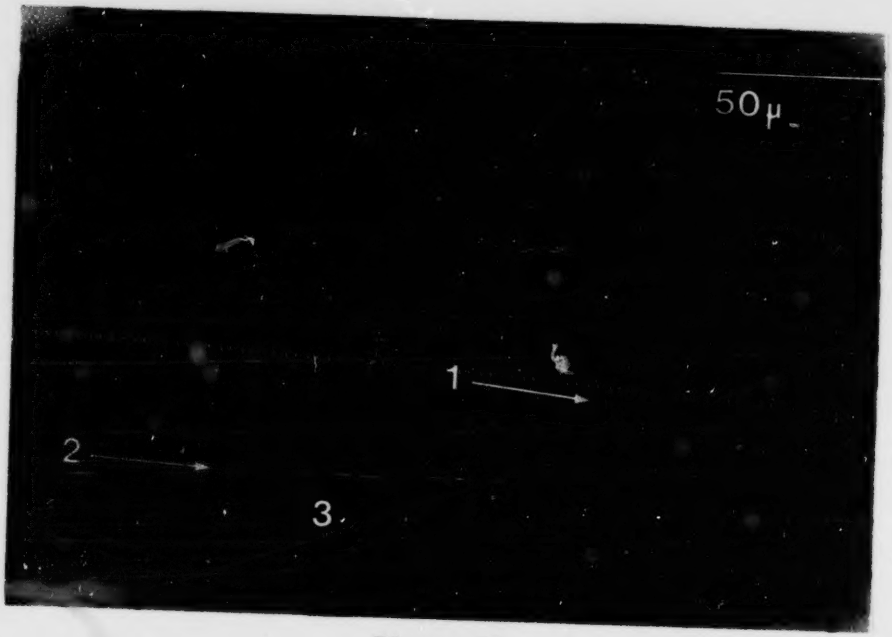

Figure 43
Subrounded inclusions of tellurobismuthite (1) and chalcopyrite - pyrrhotite (2) within pyrite (3).

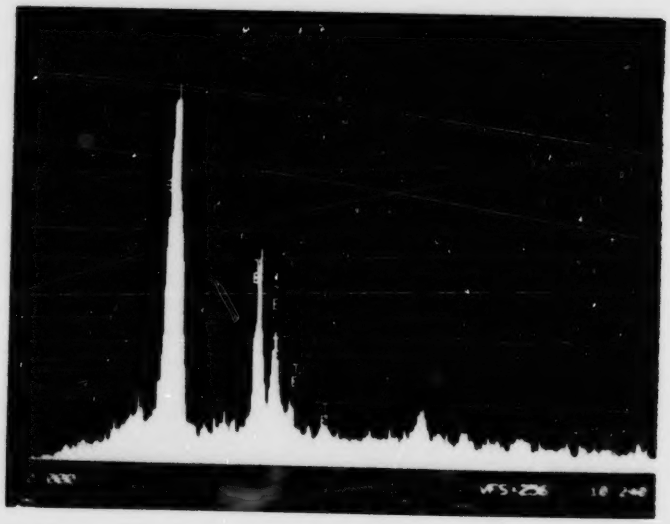

Figure 44
SEM-EDAX spectra of tellurobismuthite.

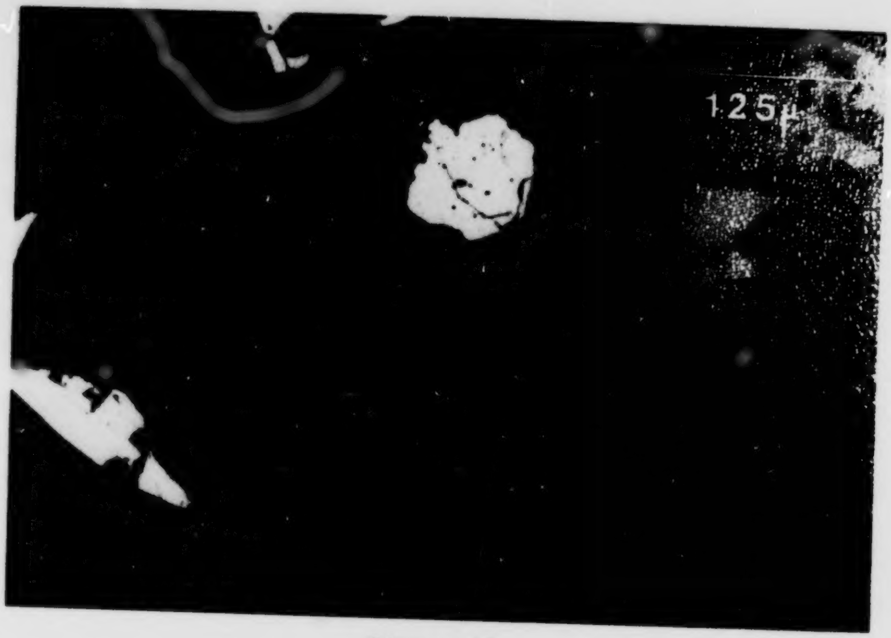

Figure 45
Angular particles of rutile and baryte commonly exhibiting grey-white colouration and whitish internal reflections.

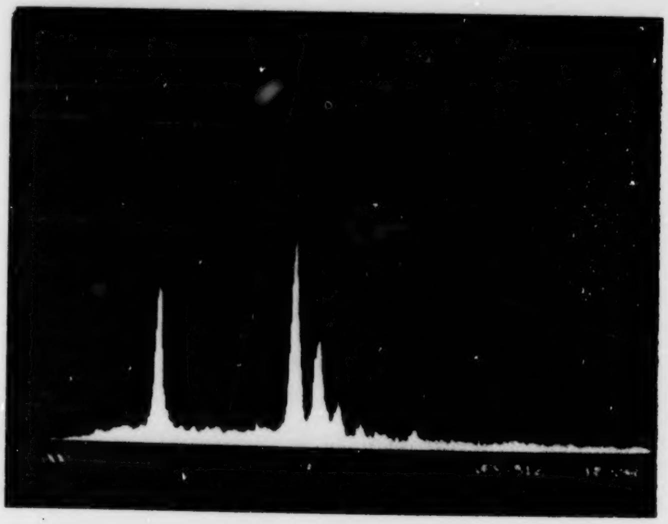

Figure 46
SEM-EDAX spectra of baryte.

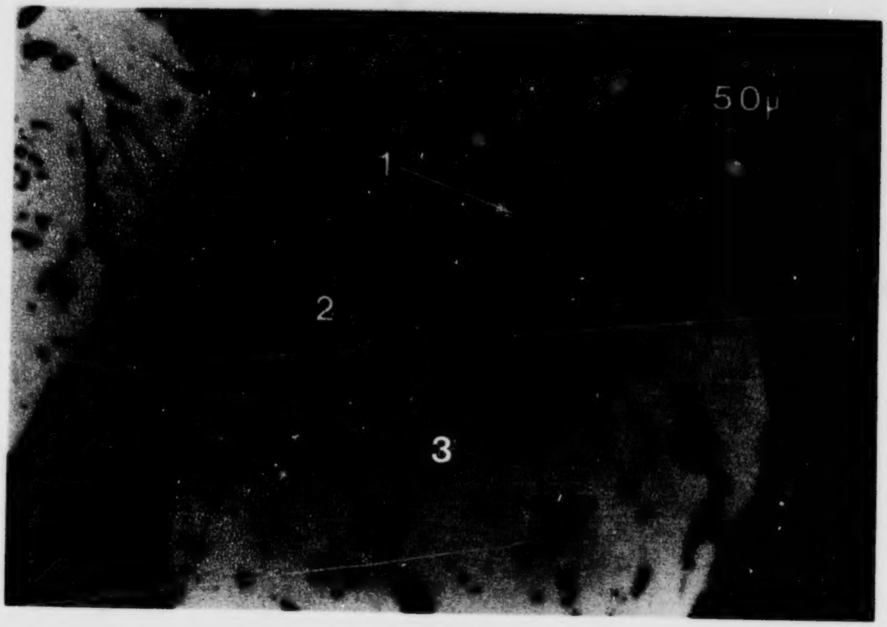

Figure 47
Minute subrounded particle of gold (1) exposed and associated with secondary iron oxide crust (2) around pyrite (3).

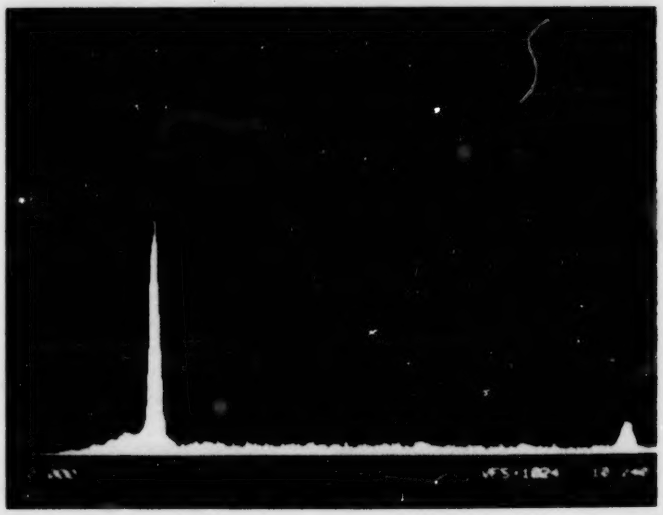

Figure 48
SEM-EDAX spectra of minute free gold particle.

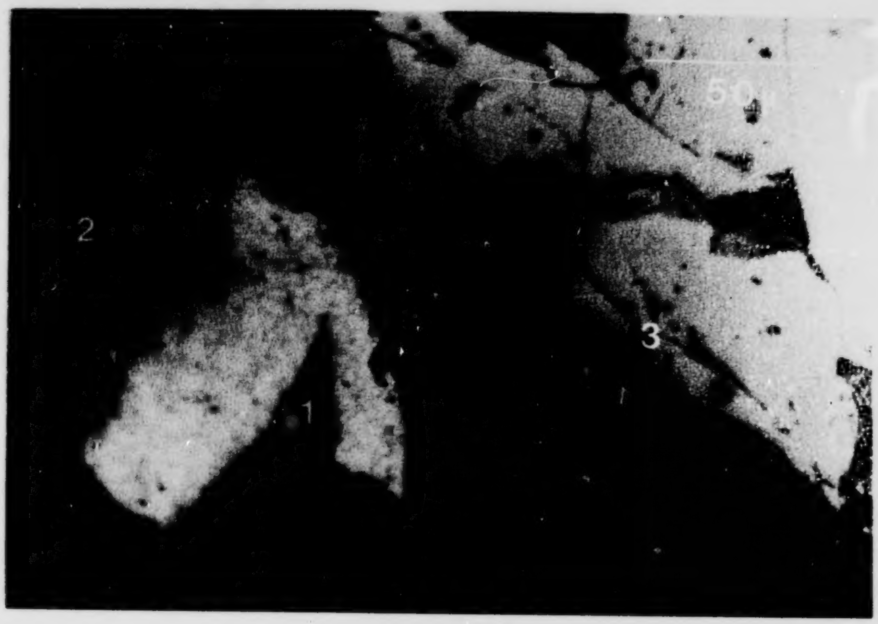

Figure 49
Angular free particle of argentiferous gold (1) exhibits minute rim of less argentiferous gold around particle (2). Large angular particle of pyrite at right (3).

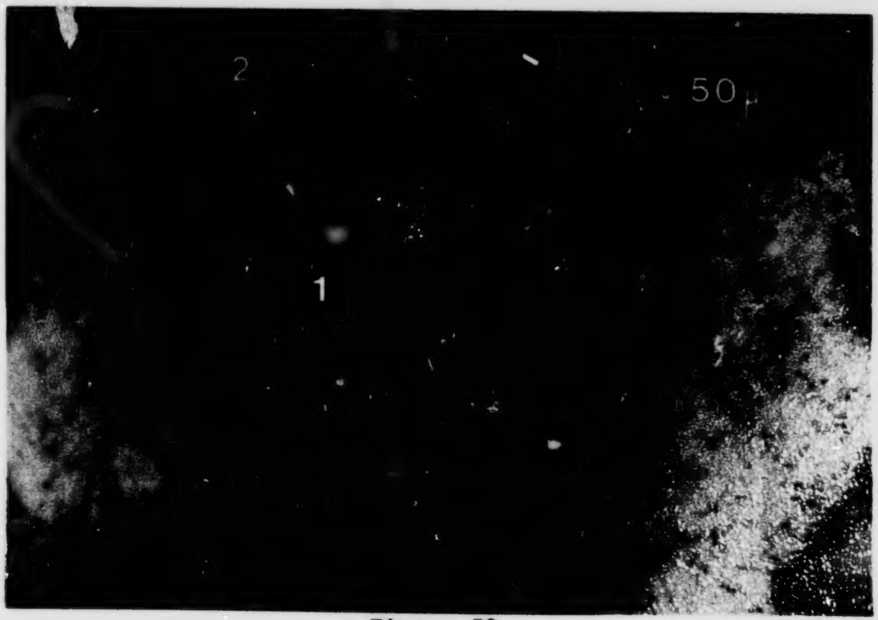

Figure 50
Closeup view of previous argentiferous gold particles (1) exhibiting discontinuous crust of gold (2). Colour and SEM spectra indicate higher fineness of gold along rim.

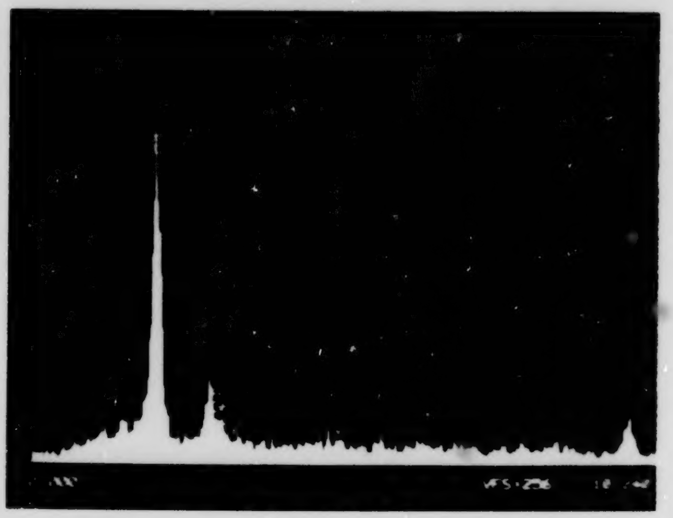

Figure 51
SEM-EDAX spectra of argentiferous core of
gold particle.

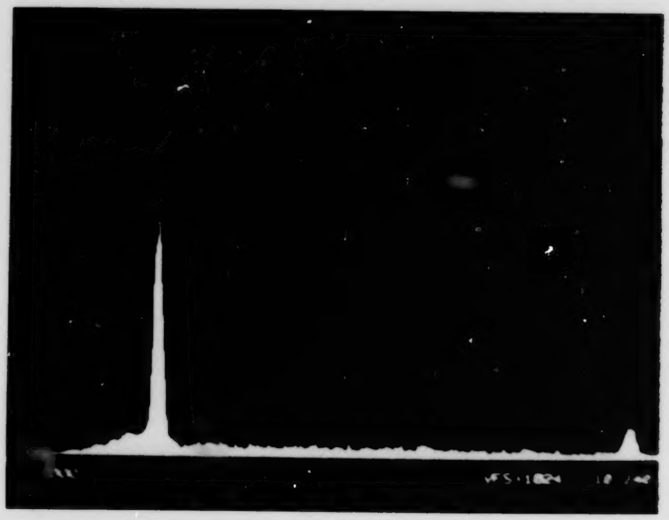

Figure 52
SEM-EDAX spectra of gold (low silver) rim
on argentiferous gold particle.

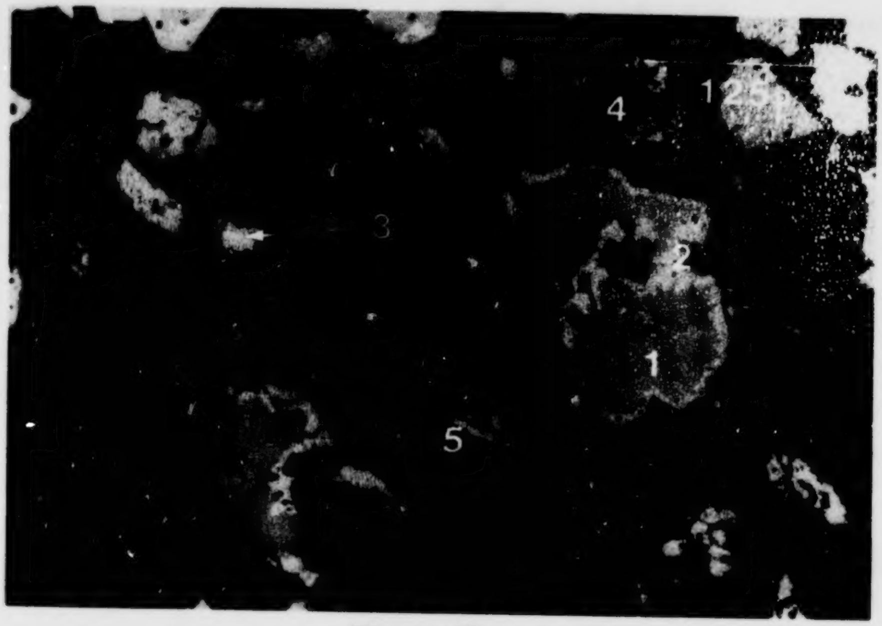

Figure 53
Homogeneous particle of titaniferous magnetite (1) commonly marginal replacement by ilmenite (2). The oxides may enclose inclusions of chalcopyrite (3). Myrmekitic intergrowths of magnetite and ilmenite (4) are present. Radiating blades of hematite (5) are common.

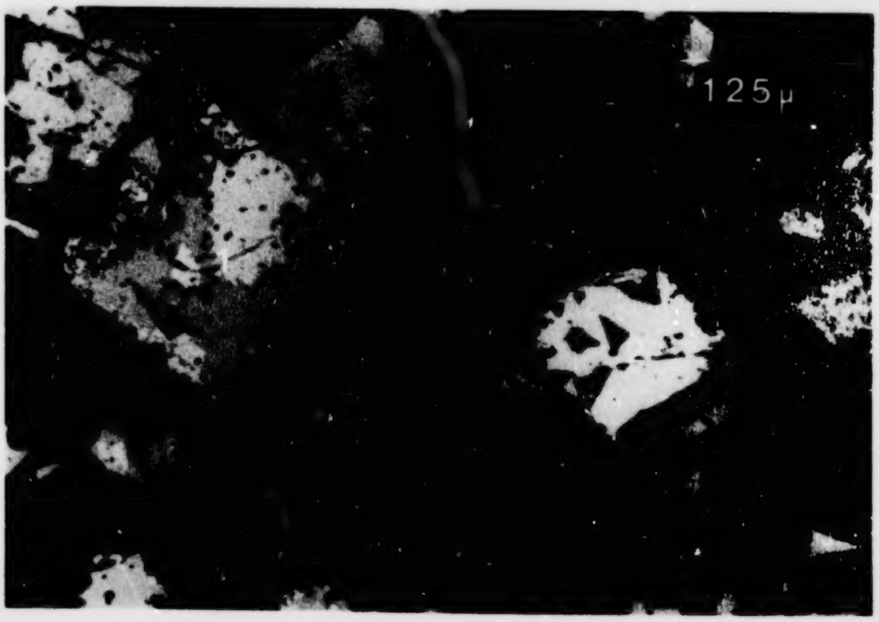

Figure 54
Intergrowths of titanomagnetite and ilmenite (1) and less common free, angular particles of pyrrhotite (2).

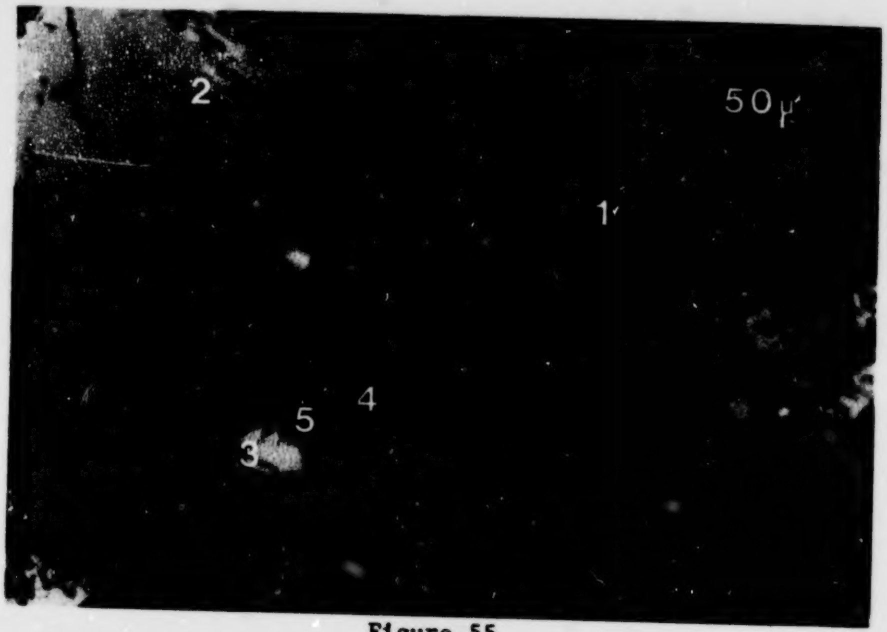

Figure 55
Myrmekitic intergrowth of magnetite and ilmenite (1), lamellae of ilmenite in titanomagnetite (2), and free particles of chalcopyrite (3) with partial replacement by chalcocite (4) and covellite (5).

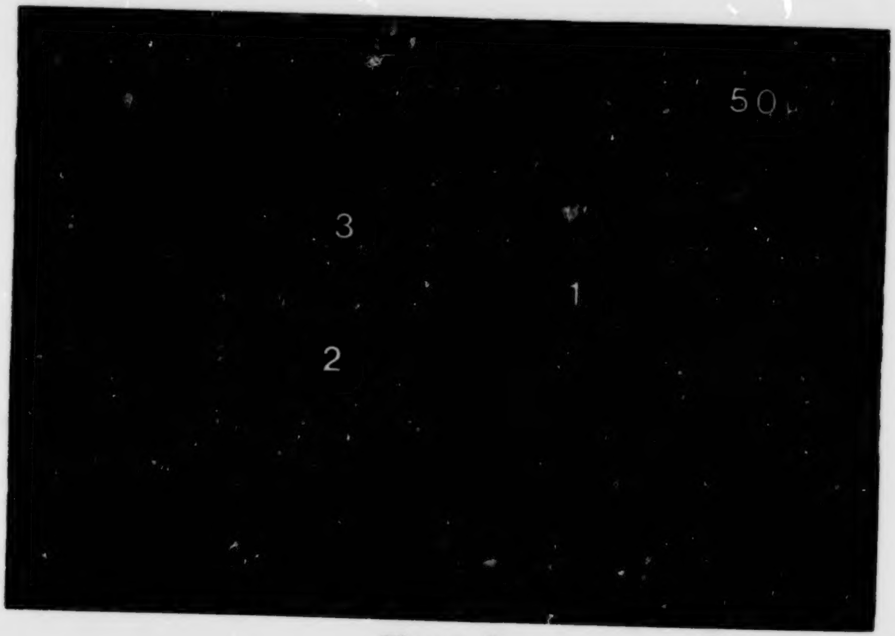

Figure 56
Lamellar to myrmekitic intergrowth of magnetite (1) and ilmenite (2) with partial replacement of magnetite by goethite and other secondary products (3).

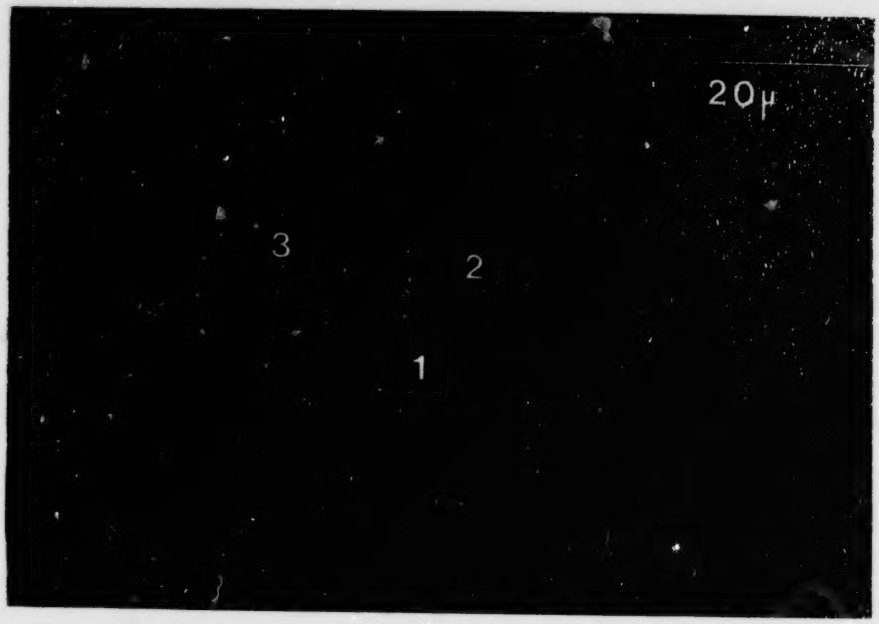

Figure 57
Inclusion of chalcopyrite (1) rimmed by replacement product of covellite (2) within magnetite - ilmenite (3).

Figure 58
Rounded inclusion of chalcopyrite (1) in lamellar intergrowth with bornite (2) contained by hematitized magnetite (3).

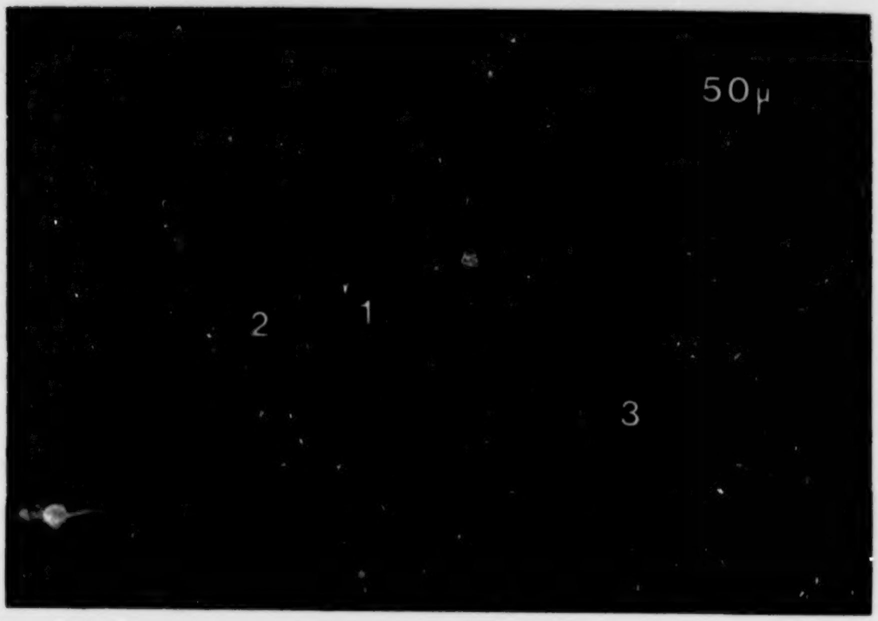

Figure 59
Minute elongate inclusions of pyrrhotite (1) with lesser rutile (2) within arsenopyrite (3).

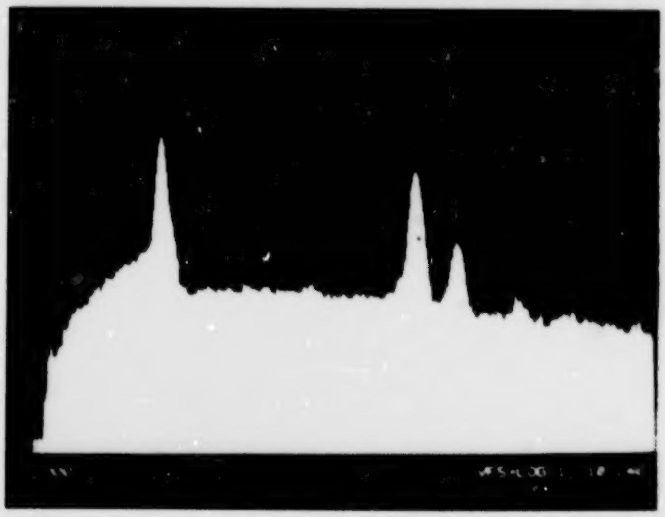

Figure 60
SEM - EDAX spectra of pyrrhotite.

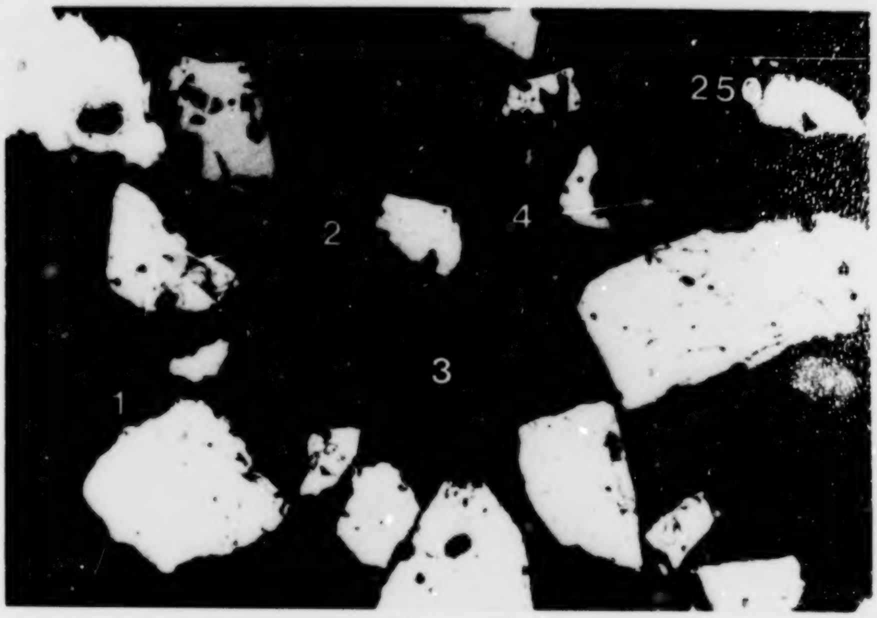

Figure 61
Angular particles of arsenopyrite (1) exhibiting weak fracturing and may contain inclusions of pyrrhotite (2). Free angular particles of pyrrhotite (3) and magnetite (4) are present.

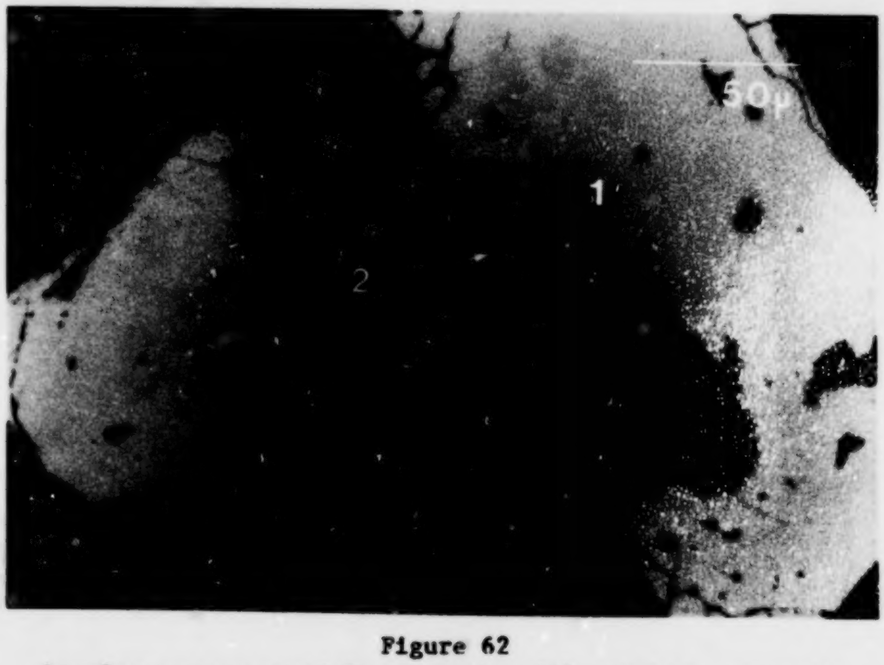

Figure 62
Angular coarse particle of pyrite (1) exhibits crust of collomorphic goethite - hematite (2). Subhedral to subangular arsenopyrite displays no fractures or alteration.

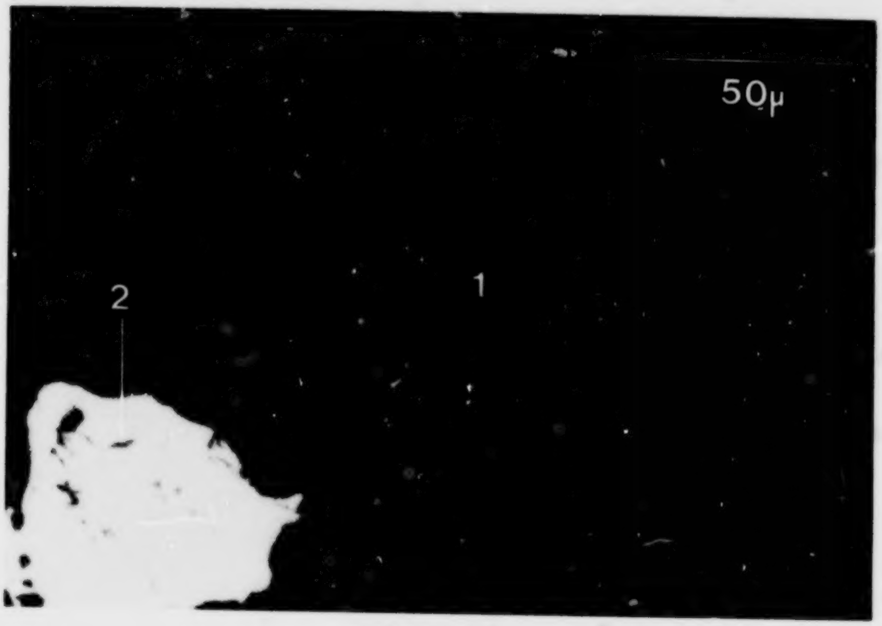

Figure 63
Aggregate particle of collomorphic goethite and secondary iron hydroxides (1) and angular particle of arsenopyrite (2).

Figure 64
Angular particle of metallic iron from contamination during grinding.

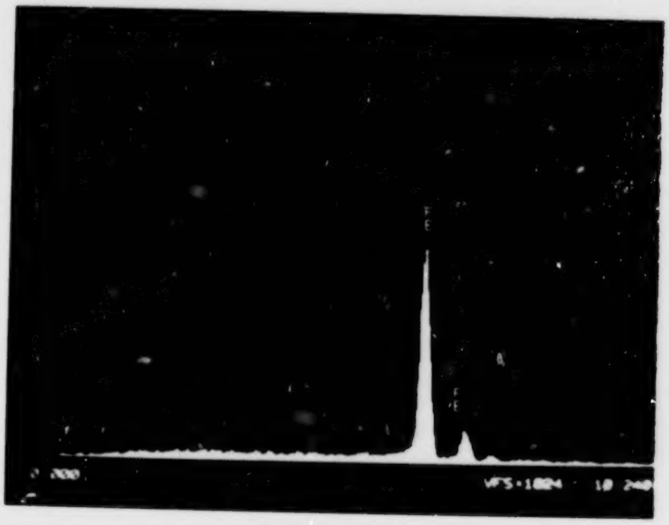

Figure 65
SEM - EDAX spectra of metallic iron fragment.

Figure 66
Inclusion of galena (1) within pyrite (2). Note the consertal grain boundary between galena and pyrite.

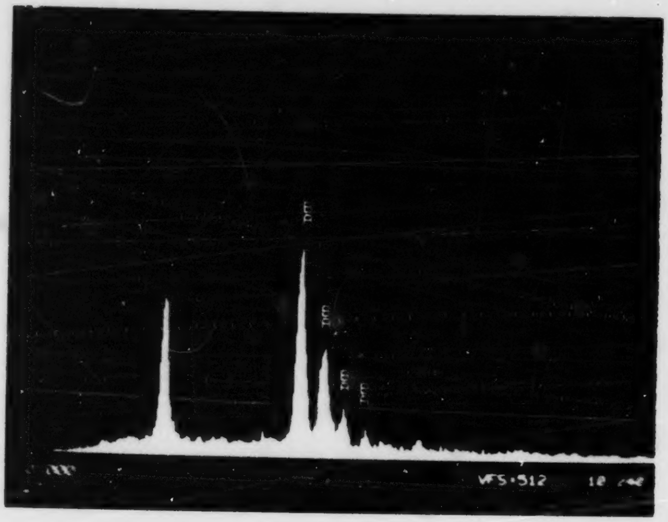

Figure 67
SEM - EDAX spectra of baryte.

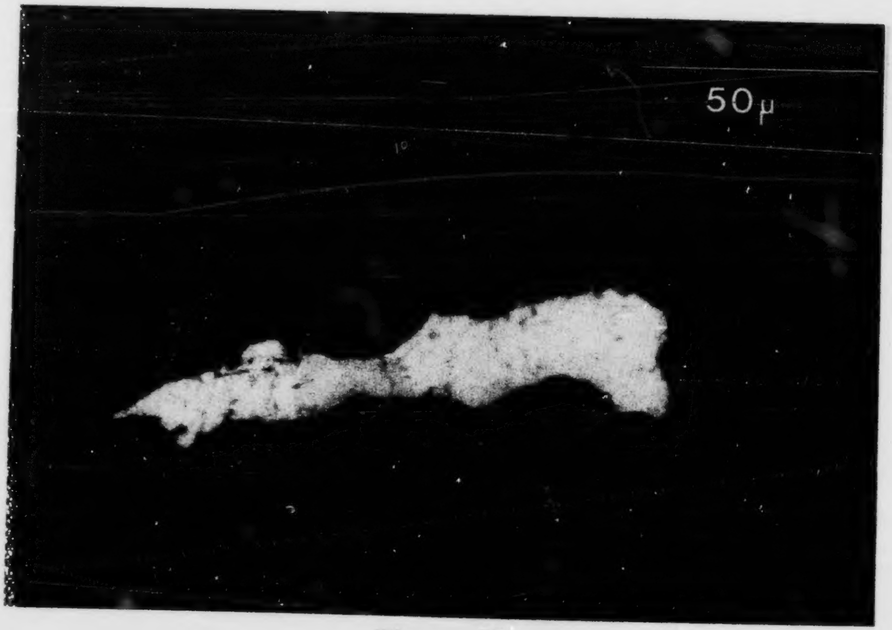

Figure 68
Elongate angular free particle of native gold.

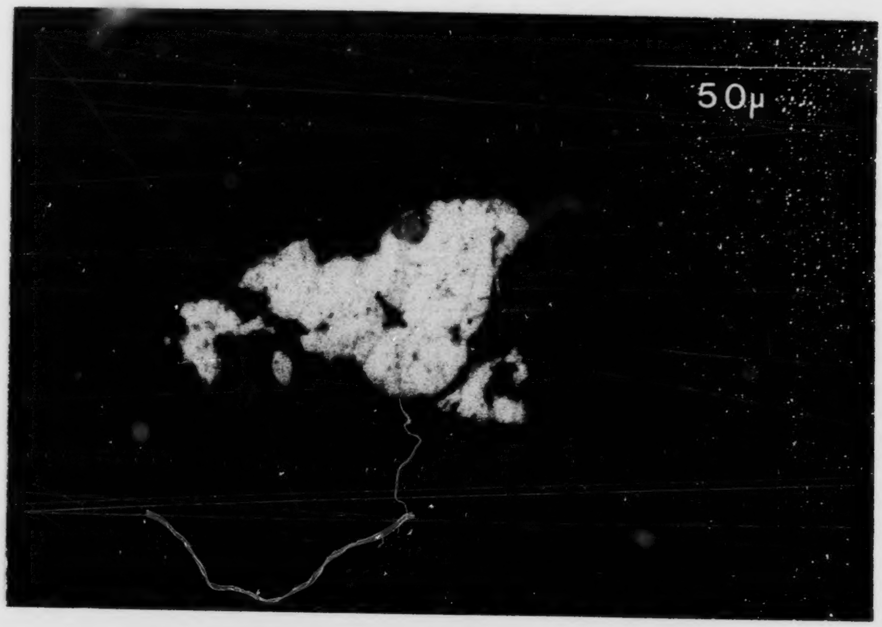

Figure 69
Irregular free particle of native gold.

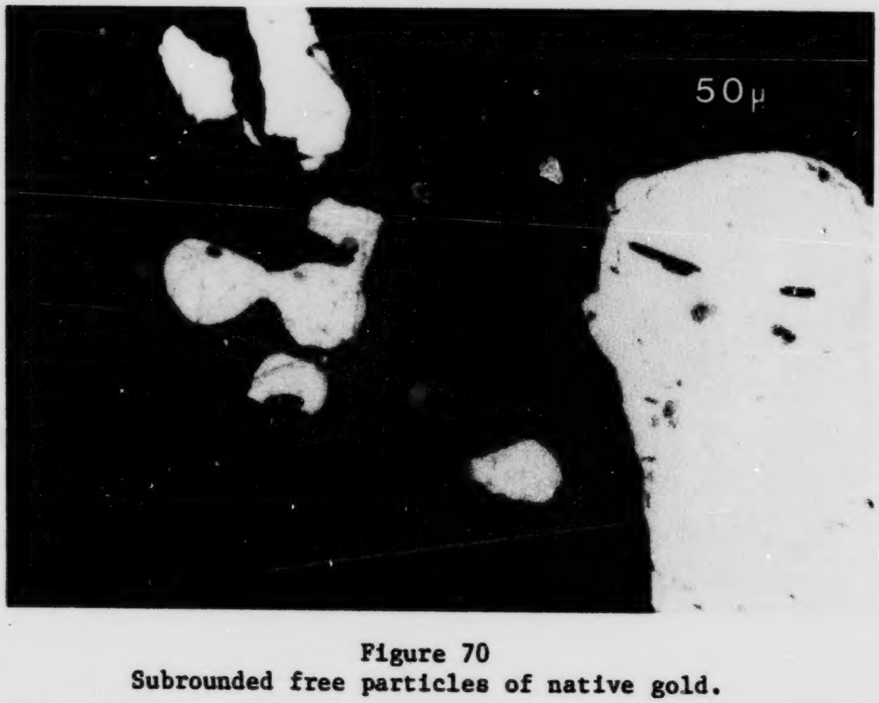

Figure 70
Subrounded free particles of native gold.

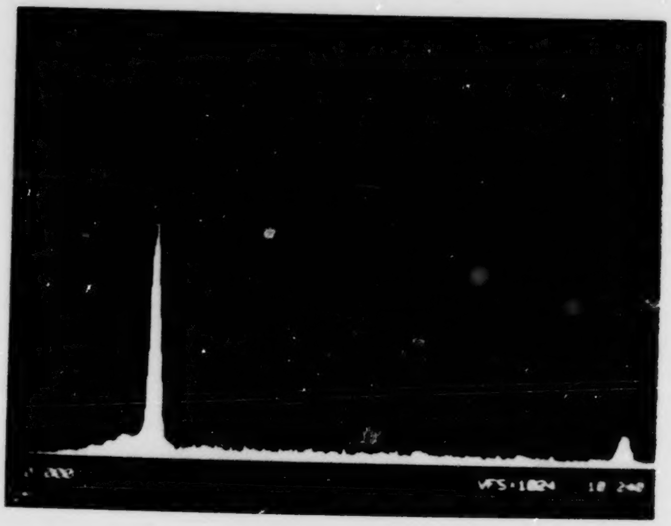

Figure 71
SEM - EDAX spectra of native gold.

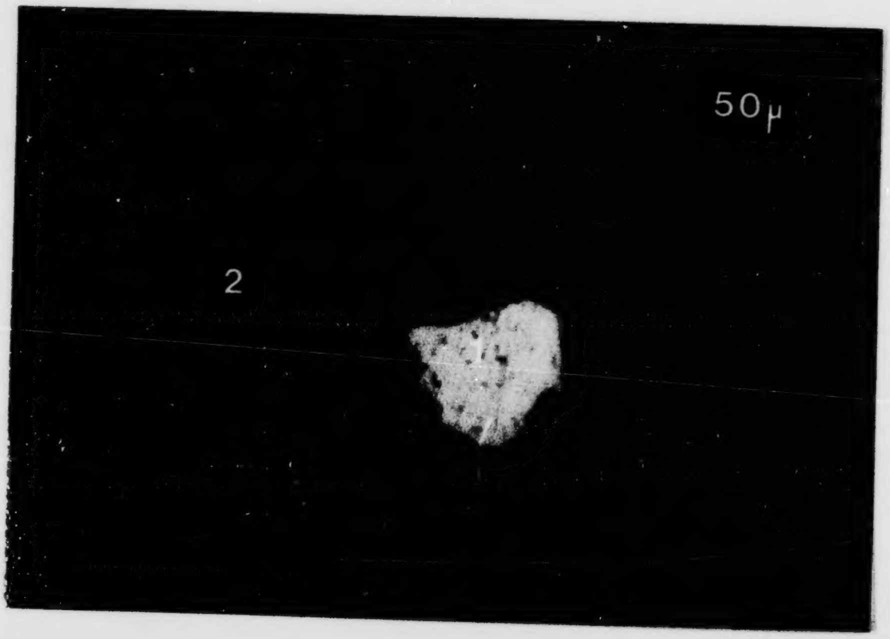

Figure 72
Subhedral rounded particle of native gold (1) attached to gangue particle (2). Note ink outline of gangue particle.

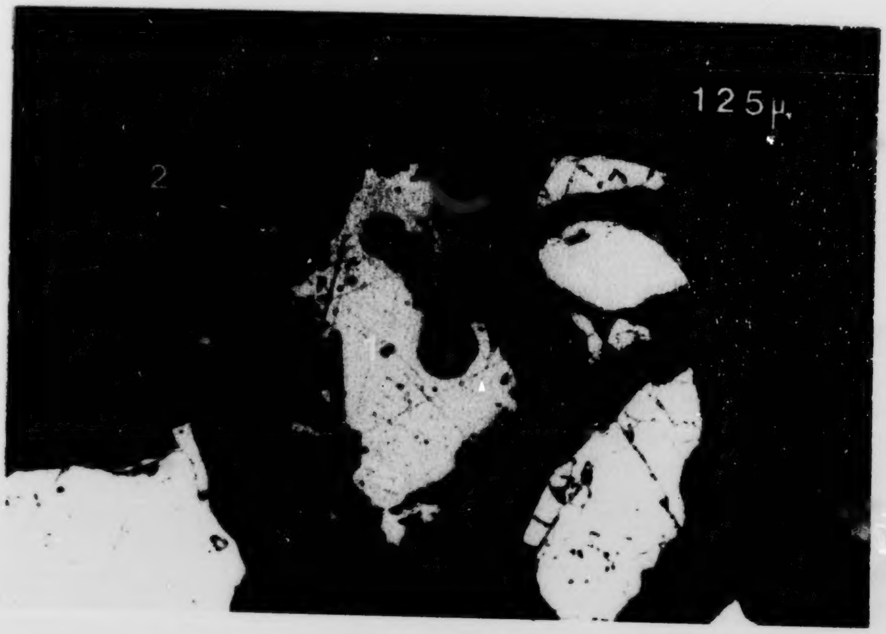

Figure 73
Irregular grain of gold (1) with secondary iron oxides (2). Minute lenticles of gold may indicate fracture filling mineralization. Several particles of arsenopyrite (3) are present.

APPENDIX III

AGITATED LEACH CYANIDATION SUMMARY

APPENDIX IV

HEAP LEACH CYANIDATION SUMMARY

CYANIDATION TEST SUMMARY

TEST NO. : Archer 1

PURPOSE : Standard Leach at pH+ of 10.5

FEED : 497.0 grams of TR2ON zone 2 (WDI NO. 85-112)
(-6 mesh material)

GRIND : 15 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 50 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 10.3 - 10.8

CONSUMPTIONS (kg/t) NaCN : 2.36 Lime : 0.81

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	480	1.73	0.83	74.1
Wash	1000	0.26	0.26	23.2
Titration	430	0.07	0.03	2.7
Residue	482.0	<0.03 *	0.00	0.0
Total		(2.25)	1.12	100.0

Au EXTRACTION : >99.0 %
ASSAYED HEAD : 0.72 g/t
CALC. HEAD : 2.25 g/t
MASS BALANCE : 312.5 %

Comments : * - Average of <0.03, and <0.03 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 2

PURPOSE : Repeat test No. 1 at higher pH+

FEED : 501.0 grams of TR2ON zone 2 (WDI NO. 85-112)
(-6 mesh material)

GRIND : 15 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 50 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 11.1 - 11.5

CONSUMPTIONS (kg/t) NaCN : 1.42 Lime : 2.09

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	430	1.26	0.54	72.0
Wash	1000	0.19	0.19	25.3
Titration	440	0.04	0.02	2.7
Residue	482.0	<0.03 *	0.00	0.0
Total		(1.50)	0.75	100.0

Au EXTRACTION : >99.0 % CALC. HEAD : 1.50 g/t
ASSAYED HEAD : 0.72 g/t MASS BALANCE : 208.3 %

Comments : * - Average of <0.03, and <0.03 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 3

PURPOSE : Standard Leach at pH+ of 10.5

FEED : 504.5 grams of Revenue Ore (WDI NO. 85-116)
(-6 mesh material)

GRIND : 10 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 45 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 10.3 - 10.8

CONSUMPTIONS (kg/t) NaCH : 1.67 Lime : 2.93

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	600	0.18	0.11	37.9
Wash	1000	0.02	0.02	6.9
Titration	200	0.008	0.00	0.0
Residue	483.0	0.33 *	0.16	55.2
Total		(0.57)	0.29	100.0

Au EXTRACTION : 44.8 %
ASSAYED HEAD : 1.15 g/t
CALC. HEAD : 0.57 g/t
MASS BALANCE : 49.7 %

Comments : * - Average of 0.39, and 0.27 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 4

PURPOSE : Repeat test No. 3 with higher pH+

FEED : 500.4 grams of Revenue Ore (WDI NO. 85-116)
(-6 mesh material)

GRIND : 10 mins at 65 % solids;

PRE-AER : none

LEACH : 24 hours; 40 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 11.5

CONSUMPTIONS (kg/t) NaCN : 1.24 Lime : 19.60

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	620	0.35	0.22	62.9
Wash	1000	0.05	0.05	14.3
Titration	200	0.008	0.00	0.0
Residue	485	0.16 *	0.08	22.8
Total		(0.70)	0.35	100.0

Au EXTRACTION : 77.1 %
ASSAYED HEAD : 1.15 g/t

CALC. HEAD : 0.70 g/t
MASS BALANCE : 60.9 %

COMMENTS : * - average of 0.28 , 0.10 , and 0.10 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 5

PURPOSE : Standard Leach at pH+ of 10.5

FEED : 502.0 grams of Casino Ore (WDI NO. 85-115)
(-6 mesh material)

GRIND : 10 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 50 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 10.3 - 10.6

CONSUMPTIONS (kg/t) NaCN : 1.35 Lime : 3.11

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	400	0.75	0.30	53.5
Wash	1000	0.09	0.09	16.1
Titration	320	0.02	0.01	1.8
Residue	408.0	0.32 *	0.16	28.6
Total		(1.12)	0.56	100.0

Au EXTRACTION : 71.4 %
ASSAYED HEAD : 1.10 g/t
CALC. HEAD : 1.12 g/t
MASS BALANCE : 101.8 %

Comments : * - Average of 0.33, and 0.30 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 6

PURPOSE : Repeat test No. 5 with higher pH+

FEED : 498.0 grams of Casino Ore (WDI NO. 85-115)
(-6 mesh material)

GRIND : 10 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 50 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 11.1 - 11.5

CONSUMPTIONS (kg/t) NaCN : 0.18 Lime : 8.57

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	405	0.80	0.32	53.3
Wash	1000	0.12	0.12	20.0
Titration	320	0.02	0.01	1.7
Residue	494.0	0.30 *	0.15	25.0
Total		(1.20)	0.60	100.0

Au EXTRACTION : 75.0 %
ASSAYED HEAD : 1.10 g/t

CALC. HEAD : 1.20 g/t
MASS BALANCE : 109.1 %

Comments : * - Average of 0.28, and 0.32 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 7

PURPOSE : Standard Leach at pH+ of 10.5

FEED : 503.0 grams of TR24N zone 3 (WDI NO. 85-113)
(-6 mesh material)

GRIND : 5 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 40 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 10.3 - 10.7

CONSUMPTIONS (kg/t) NaCN : 1.59 Lime : 7.32

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	640	0.98	0.63	75.0
Wash	1010	0.12	0.12	14.3
Titration	260	0.024	0.01	1.2
Residue	483.0	0.16 *	0.08	9.5
Total		(1.67)	0.84	100.0

Au EXTRACTION : 90.5 % CALC. HEAD : 1.67 g/t
ASSAYED HEAD : 1.33 g/t MASS BALANCE : 125.6 %

Comments : * - Average of 0.11, and 0.21 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : Archer 8

PURPOSE : Repeat test No. 7 with higher pH+

FEED : 492.3 grams of TR24N zone 3 (WDI NO. 85-113)
(-6 mesh material)

GRIND : 5 minutes; 65 % solids;

PRE-AER : none

LEACH : 24 hours; 40 % solids

REPULP : none

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Leach - NaCN - 1.00 gpl free
- Lime - pH+ 11.2 - 11.6

CONSUMPTIONS (kg/t) NaCN : 0.28 Lime : 12.07

Au MASS BALANCE

Product	WT/VOL g/ml	Au ppm	Au mg	Au Dist'n
Filtrate	620	0.98	0.61	70.1
Wash	1000	0.14	0.14	16.1
Titration	240	<0.005	0.00	0.0
Residue	480.0	0.26 *	0.12	13.8
Total		(1.77)	0.87	100.0

Au EXTRACTION : 86.2 % CALC. HEAD : 1.77 g/t
ASSAYED HEAD : 1.33 g/t MASS BALANCE : 133.1 %

Comments : * - Average of 0.25, and 0.27 g/t Au

CYANIDATION HEAP LEACH TEST SUMMARY

TEST NO. : HL-1

FEED : 833.5 kgs of TR2ON zone 2 (WDI No. 85-112)

DAYS LEACHING : 15

WASH : wash with 185 litres of water

REAGENTS : NaCN - start at 1.00 gpl , then let drop to 0.50gpl and bring level back up to 1.00 gpl 1 day before end of leach

Lime - pH: 9.9 - 10.5

CONSUMPTIONS (kg/t) NaCN : 0.79 Lime : 0.15

Au MASS BALANCE

Product	WT/VOL kg/ l	Au ppm	Au mg	Au Dist'n
Filtrate	232	0.05	12.5	2.2
Wash	185	<0.008	1.3	0.2
Carbons	0.49587	(1074.27)	532.7	93.2
Residue	833.5	(<0.03)	25.0	4.4
Total		(0.69)	571.5	100.0

Au EXTRACTION	:	95.6 %	CALC. HEAD	:	0.69	g/t
ASSAYED HEAD	:	0.71 g/t	MASS BALANCE	:	97.2	%

CYANIDATION HEAP LEACH TEST SUMMARY

TEST NO. : HL-2

FEED : 391 kgs of REVENUE (WDI No. 85-116)

DAYS LEACHING : 31

WASH : wash with 210 litres of water

REAGENTS : NaCN - start at 0.50gpl , and bring level up to 1.00 gpl 1 day before end of leach

Lime - pH+ 9.9 - 10.6

CONSUMPTIONS (kg/t) NaCN : 2.18 Lime : 0.95

Au MASS BALANCE

Product	WT/VOL kg/ l	Au ppm	Au mg	Au Dist'n
Filtrate	224	0.006	1.34	0.9
Wash	210	<0.005	1.05	0.7
Carbons	0.5235	(128.6)	67.33	45.5
Residue	386.5	(0.20)	25.0	52.9
Total		(0.38)	147.87	100.0

Au EXTRACTION : 47.1 %
ASSAYED HEAD : 1.15 g/t

CALC. HEAD : 0.38 g/t
MASS BALANCE : 33.0 %

CYANIDATION HEAP LEACH TEST SUMMARY

TEST NO. : HL-3

FEED : 890 kgs of CASINO (WDI No. 85-115)

DAYS LEACHING : 27

WASH : wash with 160 litres of water

REAGENTS : NaCN - start at 1.00 gpl , then let drop to 0.50gpl and bring level back up to 1.00 gpl 1 day before end of leach

Lime - pH+ 9.9 - 10.5

CONSUMPTIONS (kg/t) NaCN : 1.17 Lime : 0.64

Au MASS BALANCE

Product	WT/VOL kg/ l	Au ppm	Au mg	Au Dist'n
Filtrate	200	0.022	4.4	0.5
Wash	160	0.00	0.0	0.0
Carbons	0.5165	(355.31)	183.52	22.5
Residue	882.5	(0.713)	629.37	77.0
Total		(0.93)	817.29	100.0

Au EXTRACTION : 23.0 %

ASSAYED HEAD : 1.10 g/t

CALC. HEAD : 0.93 g/t

MASS BALANCE : 84.5 %

CYANIDATION HEAP LEACH TEST SUMMARY

TEST NO. : HL--4

FEED : 980 kgs of TR24N zone 3 (WDI No. 85-113)

DAYS LEACHING : 44

WASH : wash with 185 litres of water

REAGENTS : NaCN - start at 1.00 gpl , then let drop to 0.50gpl and bring level back up to 1.00 gpl 1 day before end of leach

Lime - pH+ 9.9 - 10.6

CONSUMPTIONS (kg/t) NaCN : 1.40 Lime : 0.71

Au MASS BALANCE

Product	Wt/Vol kg/ l	Au ppm	Au mg	Au Dist'n
Filtrate	210	0.04	8.4	0.6
Wash	185	0.00	0.0	0.0
Carbons	0.50803	(1086.0)	551.72	39.4
Residue	980	(0.857)	840.33	60.0
Total		(1.43)	1400.45	100.0

Au EXTRACTION : 40.0 %
ASSAYED HEAD : 1.33 g/t
CALC. HEAD : 1.43 g/t
MASS BALANCE : 107.5 %

CYANIDATION HEAP LEACH TEST SUMMARY

TEST NO. : HL-4 (REWASH)

FEED : 980 kgs of TR24N zone 3 (WDI No. 85-113)

DAYS LEACHING : 44

WASH : rewash 3-1 kg representative samples with 1.2 litres of pH+ 10.5 water

REAGENTS : NaCN - start at 1.00 gpl , then let drop to 0.50gpl and bring level back up to 1.00 gpl 1 day before end of leach

Lime - pH+ 9.9 - 10.6

CONSUMPTIONS (kg/t) NaCN : 1.40 Lime : 0.71

Au MASS BALANCE

Product	WT/VOL kg/ l	Au ppm	Au mg	Au Dist'n
Filtrate	210	0.04	8.4	0.5
Wash	(1176)	(0.053)	62.4	4.0
Carbons	0.50803	(1086.0)	551.72	34.0
Residue	980	(1.016)	995.87	61.5
Total		(1.65)	1620.19	100.0

Au EXTRACTION : 38.5 % CALC. HEAD : 1.65 g/t
ASSAYED HEAD : 1.33 g/t MASS BALANCE : 124.1 %

Volume

3

6-9006

WDI
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CONTRACT REPORT NO. 69006-01-SQ

DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY
PHASE II

VOLUME 1

A METALLURGICAL ASSESSMENT

Witteck Project No. 5209

for

Department of Supply and Services
11 Laurier St., 11C1
Hull, Quebec
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1.0 SUMMARY

This Phase II report presents the results of the metallurgical testwork performed on a sample of gold bearing material from the Antoniuk/Mt. Freegold deposit. It is a continuation of the preliminary metallurgical investigation that was carried out as Phase I of the project but provides a more detailed metallurgical examination of a typical sample from the deposit. This metallurgical evaluation was carried out in conjunction with Volume II of this Phase II report entitled "Mineralogy and Dye Penetrant Testswork".

Most of the metallurgical testwork in this Phase II testwork was performed on a sample of material from a trench sample designated 21 North (referred to as TR21N in this report). Due to the relatively low grade of the sample and the presence of free gold in the sample, an overall head grade was averaged using the calculated head grade for all the tests. This head grade was used for each test to calculate the gold mass balance. The average head grade for sample TR21N was 0.77 grams per tonne (0.022 oz/ton).

A number of agitated leach tests were performed using various parameters. It was found that a retention time of 72 hours was required to extract 95% of the gold. This required standard leaching parameters of 1.0 gpl free sodium cyanide and a pH of 10.5-11.0 with no pre-aeration or $PbNO_3$ additions. The addition of a pre-aeration stage with $PbNO_3$ prior to cyanidation produced a lower gold extraction of 70.9%.

Cyanidation column heap leach tests were completed on various sizes of Antoniuk TR21N zone 4 material. These tests consisted of duplicate tests using nominal feed sizes of 19 mm (3/4"), 38 mm (1 1/2"), 76 mm (3") and run of mine material consisting of up to 254 mm (10") material.

Average extractions of the duplicate tests were 80.1, 87.0, 68.2 and 72.8% gold respectively. Cyanide consumptions were high for these column tests, ranging from 3.43 to 11.26 kg/t NaCN, most likely due to operation at too low pH levels. The overall test results indicate that optimum results are obtained at pH 11 - 11.5 or 1 pH unit higher than most of the test runs. A larger pilot column test was completed on the same ore (305 kg of feed). This was performed on 38 mm (1 1/2") material using a higher pH level of 11.5 to lower the cyanide consumption. This test produced a 74.1% gold extraction and a significantly lower cyanide consumption of 2.04 kg/t.

Drill core samples of Antoniuk drill holes No. 21 and 24 were received at a later date. This drill core was from the sulphide and transition zones of the deposit which underly the oxide ore zone of which the sample TR21N was taken. Six column heap leaches were performed on various zones of the drill core at a nominal size of 38 mm (1 1/2"). Gold extractions for this material ranged from 28.5 to 61.1%, these variations due probably to the varying degrees of oxidation of the samples.

2.0 INTRODUCTION

In the first phase of the metallurgical and engineering assessment of the Dawson Range heap leach project, six surface bulk samples were evaluated for their potential for the recovery of their gold and silver content. The initial evaluation included mineralogical evaluation, permeability testing, bench scale agitated cyanidation leaches and pilot scale column heap leach testing. The results of the initial testwork on two of the samples from the Antoniuk deposit were very encouraging and resulted in a decision to proceed to a second phase of the study for a more detailed metallurgical and mineralogical assessment for bringing the property into production using low cost heap leaching techniques.

This TR21N represented a sample of the material from the oxide zone of the orebody. On August 6, 1986 drill core samples designated Antoniuk Holes No. 21 and 24 were received by Witteck for testing. These were samples from the transition and sulphide zones of the deposit that had been obtained during a diamond drilling program on the property. These samples were evaluated separately depending on the degree of oxidation.

Metallurgical evaluation of these ore types included bench scale cyanidation, bottle rolls and column heap leach testwork.

3.0 PROCEDURES

3.1 Sample Preparation

The remaining coarse bulk ore material from Phase I of the Antoniuk (Mt. Freegold) trench 21 north (TR21N) was used for this testwork. A representative lot was split out from the bulk sample by coning and quartering, crushing then further coning and quartering. The final sample was crushed to minus 6 mesh and riffled into 500 gram lots to be used for bench scale leach tests and head analysis. The half of the bulk sample that was not crushed to minus 6 mesh was a nominal three-quarter inch size. This material was representatively riffled for the bottle roll tests. After being split the samples were crushed separately to produce feed sizes of 6 mm (1/4"), 9.5 mm (3/8"), 12.7 mm (1/2") and 19.5 mm (3/4"). The remainder of the bulk sample was used for the various sized column heap leach test. Each feed sample was carefully and as representatively as possible split from the bulk sample to produce duplicate lots of 19 mm (3/4"), 38.1 mm (1 1/2"), 76 mm (3") inch size and run of mine feed materials. A pilot column heap leach feed of 38 mm (1 1/2") (305 kg) material was also taken from this bulk sample.

The drill core received for this testwork was all crushed to a nominal 38 mm (1 1/2") size. Six column heap leach tests were outlined for this core. Various zones of the drill core were combined to be used as feed for the column work. The core was combined as follows:

HL-13	Antoniuk	1-21 oxide
		2-21 oxide
		11-24 oxide I Transition I (1/2)

HL-14	Antoniuk	12-24	hypogene I (1/2)
		13-24	hypogene I
		14-24	hypogene I
		15-24	hypogene I
		16-24	hypogene I
HL-15	Antoniuk	11-24	transition I (1/2)
		12-24	transition I (1/2)
		17-24	transition II (1/2)
		18-24	transition II (1/2)
HL-16	Antoniuk	17-24	hypogene (1/2)
		18-24	hypogene II (1/2)
		19-24	hypogene II
		20-24	hypogene II
HL-17	Antoniuk	6-21	hypogene I (1/2)
		7-21	hypogene I
		8-21	hypogene I
		9-21	hypogene I
		10-21	hypogene I
HL-18	Antoniuk	3-21	transition I
		4-21	transition I
		5-21	taansition I
		6-21	transition 1 (1/2)

3.2 Head Analysis

Two bulk head samples along with a head sample from each of the TR21N column heap leaches were used to perform metallics assays. The +150 mesh fraction was all fire assayed and the -150 mesh fraction was assayed in duplicate. A metallics assay was performed because of free gold present in the ore.

All gold analyses were performed using fire assay, techniques with an atomic adsorption "finish". The gold head obtained by the fire assay procedure was dissolved in aqua regia and subjected to atomic adsorption analysis for the quantitative determination of the gold content.

3.3 Cyanidation

3.3.1 Agitated Leaching

The feed used for agitated cyanidation testwork was ground in a steel ball mill at 65% solids. After grinding, the slurry was washed into a 2 litre Pyrex leaching vessel, allowed to settle and the clear solution decanted to produce the desired slurry density of 40% solids. The slurry was agitated with a three bladed plastic impellor. All tests were dosed with 1.0 gpl sodium cyanide after the pH was stabilized at the desired level. Two of the tests used a pre-aeration stage with an addition of $PbNO_3$ prior to the cyanide addition. The slurry was filtered and repulped with fresh tap water before cyanidation when the pre-aeration staged was used. The pH was maintained at the desired level by adding lime prior to cyanide additions. The free cyanide concentrations were monitored throughout the tests by sampling and titrating the solutions using a standard $AgNO_3$ solution. Titration solutions were combined and retained for gold analysis.

At the end of each leaching period, the slurry was vacuum filtered and the cake was displacement washed with twice the dry solids weight of tap water. The volumes of filtrate, wash and titration solutions were recorded and each solution was submitted for gold analysis. The cake was dried, weighed and a metallics assay was performed on half of the residue.

3.3.2 Bottle Roll Leaching

Bottle roll cyanidation tests were performed in duplicate on samples of 6 mm (1/4"), 9.5 mm (3/8"), 12.7 mm (1/2") and 19.5 mm (3/4") size fractions. The feed was added to the plastic bottles and water was added to produce a slurry density of 50% solids. Lime was then added to obtain the desired pH level and the bottle rolled for approximately one hour. The pH was then checked, brought up to the desired level again and then cyanidation commenced. Sodium cyanide and pH levels were monitored and maintained at the desired levels throughout the tests using the same procedures employed for the agitated leach tests.

3.3.3 Column Heap Leach Testing

The feed for each column heap leach was loaded into their respective columns. The columns were mounted above a reservoir where the pregnant solution was allowed to collect. A submersible pump was installed in this reservoir to pump the pregnant solution through a carbon column to strip the gold from solution. Barren solution emerging from the carbon column was recycled back to the top of the column which percolated through the ore in the column to the reservoir, completing this closed circuit system. Sodium cyanide and pH levels were monitored and maintained using the same procedure as described for the agitated leaches. Water was added bi-weekly to compensate for evaporation and to maintain solution volumes. The solution flow rate was maintained at 0.0025 USGPM/ft^2 throughout each test.

Solutions and gold loaded activated carbon samples were taken periodically and analysed for gold. From these results, a gold extraction curve was established.

After the solution flow to the columns was stopped, the columns were allowed to drain and were washed with fresh water.

A sample of the final solution and wash water was saved and submitted for gold analysis. When washing was completed, the column was emptied and the residue was air dried. Two representative 300 gram samples were carefully split from the residue and a metallics assay was performed on each.

4.0 RESULTS

Based upon the testwork performed, the following results are summarized.

4.1 Head Analysis

The following table (Table 1) shows the results of the metallic assays performed on two head samples of the TR21N zone 4 ore.

Table 1

Head Assays

Head A

Mesh	Weight (g)	Au (g/t)	Au (mg)
+150	0.59	44.2	0.026
-150	246.10	(0.84; 0.62)	0.180
Total	246.69	0.84	0.206

Head B

Mesh	Weight (g)	Au (g/t)	Au (mg)
+150	16.90	6.81	0.115
-150	234.10	(0.69; 0.49)	0.138
Total	251.00	1.01	0.253

These assay produced an average head assay of 0.93 g/t Au (0.027 oz/t).

One sample of approximately 200 grams was as respresentatively as possible split out from each of the TR21N heap leach feeds. The following table (Table 2) shows these metallic assay results in detail.

Table 2

Column Heap Leach Head Assays

Column Heap Leach Test No.	Mesh	Weight (g)	Au (g/t)	Au (mg)
5	+150	30.1	2.04	0.061
	-150	184.5	(0.98; 0.87)	0.171
	Total	214.6	1.08	0.232
6	+150	16.25	2.52	0.041
	-150	218.10	(0.68; 0.87)	0.132
	Total	234.35	0.74	0.173
7	+150	30.06	0.83	0.025
	-150	133.80	(0.65; 0.74)	0.093
	Total	163.86	0.72	0.118
8	+150	30.11	2.79	0.084
	-150	142.20	(0.63; 0.37)	0.155
	Total	172.31	0.90	0.155
9	+150	30.12	0.75	0.023
	-150	175.00	(0.43; 0.52)	0.083
	Total	205.12	0.44	0.096

Column Heap Leach Test No.	Mesh	Weight (g)	Au (g/t)	Au (mg)
10	+150	30.51	2.02	0.062
	-150	696.00	(0.37; 0.34)	0.247
	Total	762.51	0.43	0.309
11	+150	22.23	0.25	0.006
	-150	416.90	(0.87; 0.49)	0.283
	Total	439.13	0.66	0.289
12	+150	30.12	0.75	0.023
	-150	175.00	(0.43; 0.52)	0.083
	Total	205.12	0.52	0.106

Difficulty was encountered in obtaining a representative head sample from each heap leach because of the size of material each sample had to be split out from. Since there was a wide variation in calculated heads, the mass balances for all leach tests were calculated using an average calculated head from all the tests.

4.2 Agitated Leach Tests

Six agitated leach tests were completed on the TR21N material along with two tests on the TR18N ore. All tests were run at a pH of 10.5-11.0 and varying such parameters as leach time, grind and addition of $PbNO_3$ with a pre-aeration stage. The following table (Table 3) summarizes the results of all the agitated leach tests completed. The mass balances for these tests are based on an average calculated head from all cyanidation testwork. Leach test summary sheets are provided in Appendix I.

Table 3

Agitated Leach Test Summaries

Test No.	Feed	%-200 mesh	Leach time (hrs)	Pre-aer'n Time (hrs)	Consumptions kg/t			Res -Due g/t	Calc Head g/t	Mass Bal. %	Extrn %
					NaCN	Lime	PbNO				
9	TR21N	81.3	48	-	2.31	5.43	-	0.16	0.72	91.1	77.8
10	"	78.6	48	-	2.18	5.75	-	0.12	0.77	97.5	84.2
13	TR21N	95.0	72	-	4.31	4.73	-	0.03	1.17	148.1	98.3
14	"	95.0	72	-	3.22	4.94	-	0.03	0.39	49.4	94.4
15	"	95.0	72	8	2.45	5.29	0.25	0.17	0.62	78.5	74.2
16	"	95.0	72	8	2.18	5.59	0.25	0.25	0.78	98.7	67.5
11	TR18N	83.5	48	-	2.06	6.33	-	0.23	3.67	72.0	94.0
12	"	81.5	48	-	2.01	6.20	-	0.18	4.80	94.1	96.2

Residue metallic assay results are given at the bottom of their respective leach summary sheet in Appendix I.

4.3

Seven bottle rolls cyanidation tests were completed on the TR21N composite. All tests were leached for 72 hours at a pH of 10.5-11.0 with 1.0 gpl free NaCN. Mass balances were also calculated based on the average calculated head from all cyanidation tests. This average calculated head was 0.79 g/t Au. The following table (Table 4) summarizes the results from these tests. A summary leach sheet for each test is shown in Appendix I.

Table 4

Bottle Rolls Cyanidation Summaries

Test No.	Feed Size inch.	Consumptions kg/t NaCN	Consumptions kg/t Lime	Res Due g/t	Calc Head g/t	Mass Bal. %	Extr'n %
17	3/4	1.50	3.97	0.77	1.36	172.2	48.1
21	3/4	0.85	4.03	0.28	0.79	100.0	65.8
22	1/2	1.34	5.13	0.28	0.78	98.7	64.1
19	3/8	1.37	4.09	0.28	0.96	121.5	70.8
23	3/8	1.18	5.21	0.20	0.61	77.2	67.8
20	1/4	1.61	4.86	0.27	0.92	116.5	70.3
24	1/4	1.41	5.31	0.19	0.71	89.9	72.9

Residue metallic assay results are shown at the bottom of their respective leach summary sheet in Appendix I.

4.4 Column Heap Leaching

A total of 15 cyanidation column heap leach tests were completed on TR21N and Antoniuk drill holes 21 and 24 (tests 13 - 19). All tests were run at a pH of 10.5-11.0 and a 0.80 gpl free sodium cyanide level. The only exception was Test No. 19, which was leached at a higher pH of 11.0-11.5. A head sample was not split out from any of the drill core zones, therefore, a mass balance was not calculated for these tests. The following table (Table 5) summarizes the column heap leach results. A summary leach sheet for each test is supplied in Appendix II. All TR21N Mass Balances are based on an overall cyanidation calculated head average of 0.77 g/t Au.

Table 5

Column Heap Leach Summaries

				Consumption		Au			
Test No.	Feed	Feed Size Inch.	Leach Time Days	NaCN kg/t	Lime kg/t	Res g/t	Calc Head g/t	Mass Bal. %	Extr'n %
5	TR21N	3/4	61	8.94	0.56	0.19	0.60	75.9	68.4
6	"	3/4	61	11.26	1.76	0.09	1.09	138.0	91.7
7	"	1 1/2	69	6.13	2.31	0.115	0.87	110.1	86.8
8	"	1 1/2	69	7.10	2.54	0.11	0.86	108.9	87.1
9	"	3	90	3.43	0.90	0.183	0.56	70.9	67.9
10	"	3	90	3.47	1.47	0.184	0.59	78.5	68.5
11	"	rom	90	6.61	2.22	0.172	0.52	65.8	67.1
12	"	rom	90	6.32	2.97	0.165	0.76	96.2	78.4
13	Ant.oxide	1 1/2	82	4.63	1.74	0.77	1.98	-	61.1
14	Ant.Hyp.I	1 1/2	90	4.01	0.56	3.08	4.31	-	28.5
15	Ant.Tran.I,II	1 1/2	90	5.06	1.51	3.13	7.46	-	58.0
16	Ant.Hypo.I,II	1 1/2	82	4.61	1.03	1.00	1.67	-	40.1
17	Ant.Hypo.I	1 1/2	82	3.64	0.85	0.60	0.97	-	38.4
18	Ant.Tran.I	1 1/2	82	5.01	1.81	0.58	0.89	-	35.6
19	TR21N	1 1/2	86	2.04	3.90	0.19	0.72	78.3	74.1

4.4.1 Column Leach Effluent Analysis

Final samples of effluents from heap leaches No. 5, 6, 7 and 8 were analysed for various elements. Table 6 presents the results of this analysis.

Table 6

Effluent Analysis (ppm)

Test No.	Cu	Fe	Zn	As	Sb	Ni	CN(T)
5	2.53	1.78	1.03	8.91	0.30	0.11	255
6	2.47	2.59	1.65	23.1	0.30	0.10	260
7	4.52	3.02	3.27	-	-	0.13	218
8	3.74	2.20	3.59	-	-	0.17	187

4.4.2 Heap Leaching, Au Extraction vs Time

A periodic changing of the carbons and taking solution samples produced an extraction curve for each column leach test. Table 7 presents the extraction values as total milligrams, and per cent gold extracted vs time. Graphs of these results are provided in figures 1, 2, 3 and 4.

Table 7

HEADING

Column Heap Leach No. 5 (TR21N) (19 mm (3/4") Ore)			Column Heap Leach No. 6 (TR21N) (19 mm (3/4") Ore)		
Time	Cumulative Au		Time	Cumulative Au	
(days)	(mg)	%	(days)	(mg)	(%)
3	2.52	34.3	3	3.35	27.5
8	4.33	59.0	8	6.63	54.5
17	4.65	63.4	17	9.51	78.2
36	5.83	79.4	36	10.55	86.8
61	5.02	68.4	61	11.15	91.7

Column Heap Leach No. 7 (TR21N)
(38 mm (1 1/2") Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
3	8.02	35.0
8	12.02	52.04
17	16.19	70.6
36	19.13	84.2
61	19.69	85.9
69	19.89	86.8

Column Heap Leach No. 8 (TR21N)
(38 mm (1 1/2") Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
3	6.69	33.0
8	10.76	50.3
17	13.76	67.8
36	16.80	82.8
61	17.35	85.5
69	17.69	87.1

Column Heap Leach No. 9 (TR21N)
(76 mm (3") Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
3	4.55	13.1
8	5.89	17.0
17	10.32	30.6
36	17.30	50.0
62	19.69	56.7
91	23.54	67.9

Column Heap Leach No. 10
(76 mm (3") Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
3	5.35	13.7
8	8.43	21.7
17	11.22	28.8
36	18.84	48.4
62	22.90	58.8
91	25.14	64.6

Column Heap Leach No.11 (TR21N)
(Run of Mine Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
2	5.15	9.8
7	9.86	18.7
16	18.24	34.5
35	28.61	54.2
60	30.76	58.3
90	35.45	67.1

Column Heap Leach No. 12
(Run of Mine Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
2	14.77	18.2
7	19.31	23.8
16	34.09	42.0
35	52.14	64.3
60	57.01	70.3
90	63.55	78.4

Column
Heap Leach No.15
(Antoniuk Trans)

Time	Cumulative Au	
(days)	(mg)	(%)
7	35.90	12.7
22	81.56	28.9
44	126.03	44.6
59	139.48	49.3
90	164.07	58.0

Column
Heap Leach No.16
(Antoniuk Hypo)

Time	Cumulative Au	
(days)	(mg)	(%)
7	10.25	17.3
22	13.55	22.9
44	19.63	33.2
59	22.08	37.3
82	23.71	40.1

Column
Heap Leach No.17
(Antoniuk Hypo)

Time	Cumulative Au	
(days)	(mg)	(%)
7	6.90	15.7
22	9.80	22.4
44	14.10	32.2
59	15.22	34.7
82	16.83	38.4

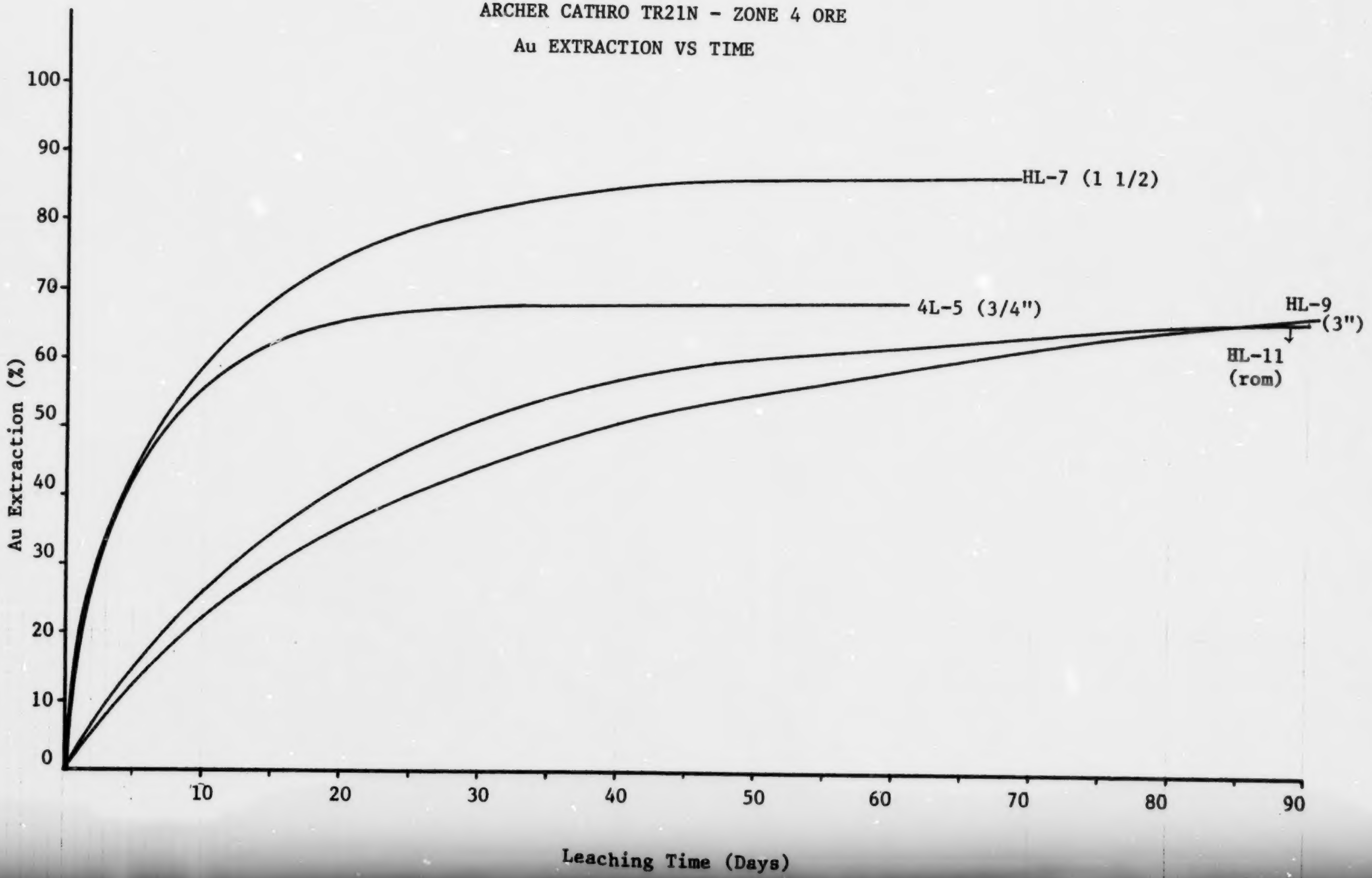
Column
Heap Leach No. 18
(Antoniuk Trans)

Time	Cumulative Au	
(days)	(mg)	(%)
7	6.17	18.4
22	8.17	24.4
44	10.98	32.8
59	11.48	34.3
82	11.91	35.6

Column
Heap Leach No.19
(38 mm (1 1/2")) TR21N Ore)

Time	Cumulative Au	
(days)	(mg)	(%)
7	65.47	29.9
22	118.81	54.3
44	150.05	68.6
58	150.58	68.9
79	158.34	72.4
86	162.09	74.1

Figure 1


ARCHER CATHRO TR21N - ZONE 4 ORE

Au EXTRACTION VS TIME

Figure 2

ARCHER CATHRO TR21N - Zone 4 Ore

Au Extraction Vs Time

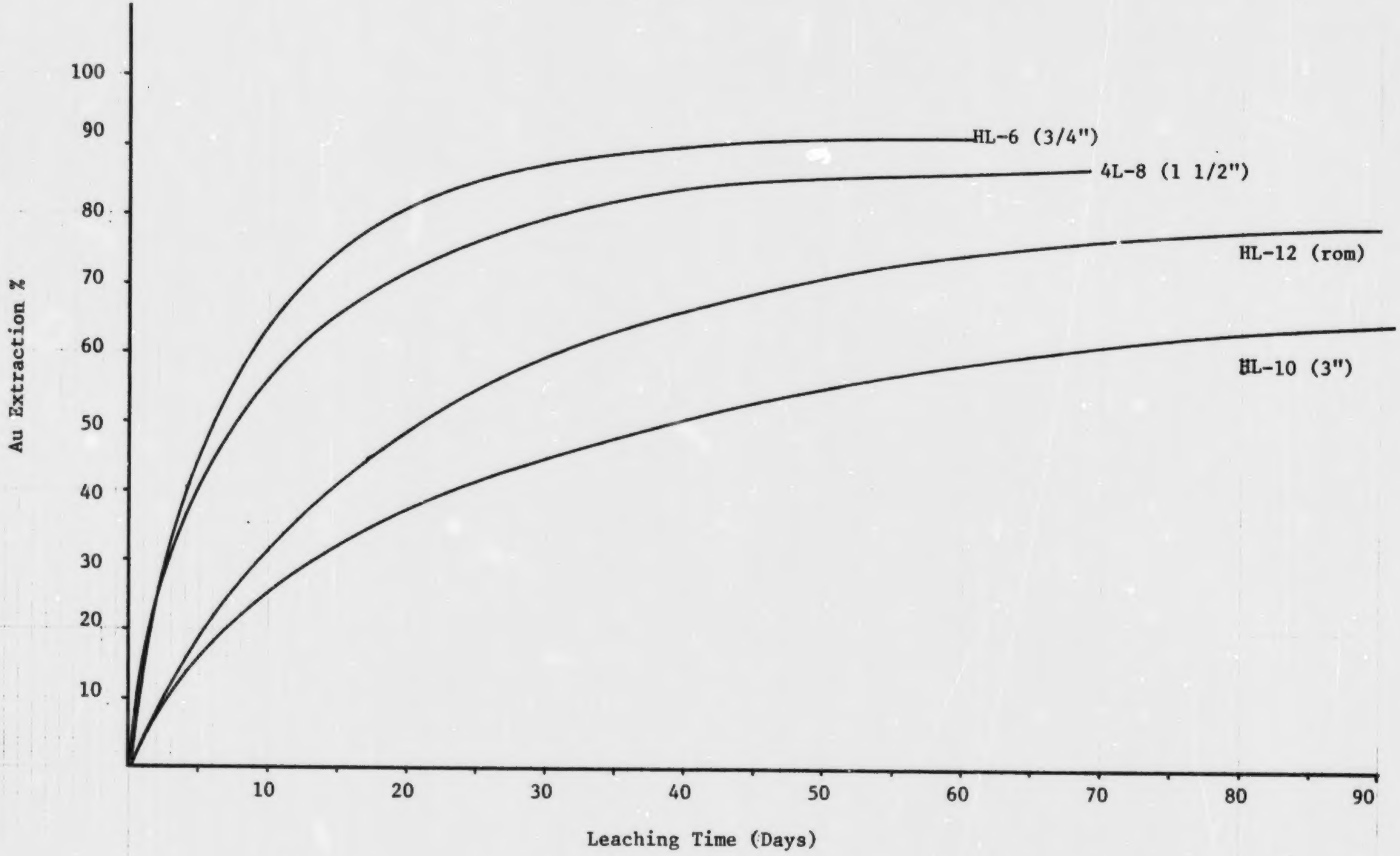


Figure 3

ARCHER CATHRO ANTONIUK DPH 21 and 24

Au Extraction Vs Time

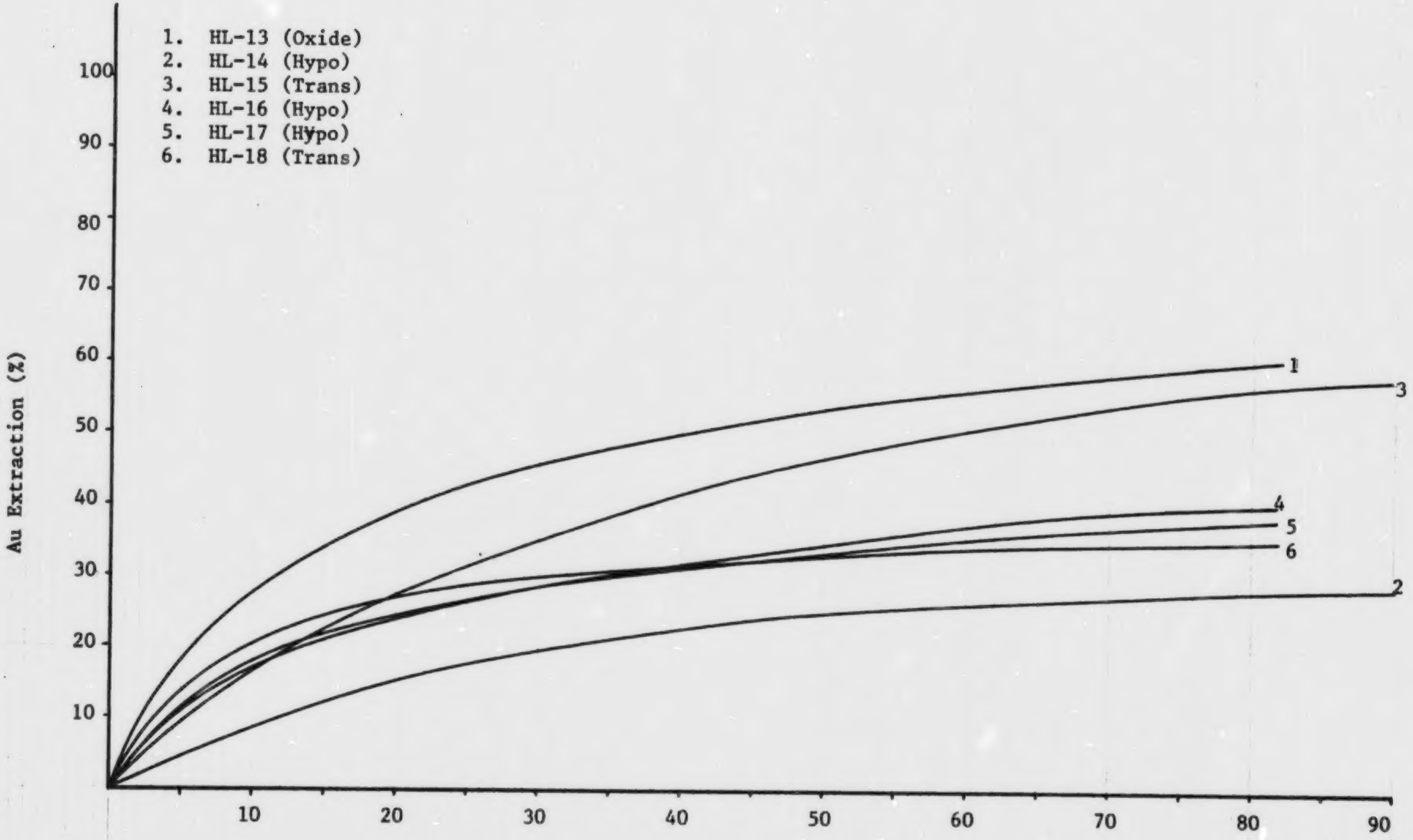
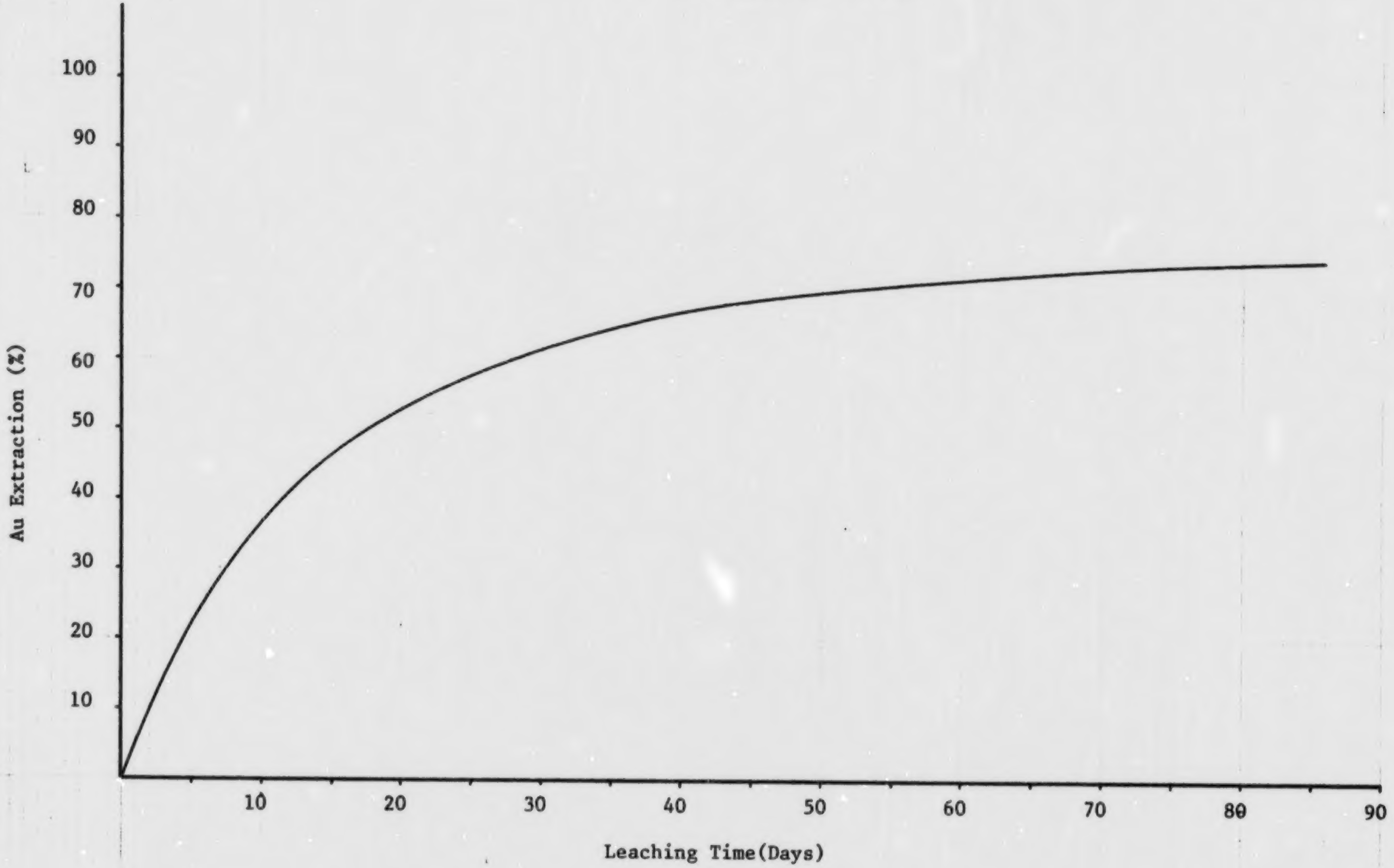


Figure 4

ARCHER CATHRO TR21N Zone 4

Heap Leach No. 19 (1 1/2" Material)

Au Extraction Vs Time



5.0 CONCLUSIONS

1.0 Fine grinding of the TR21N sample to 90% passing 200 mesh (TR21N ore), followed by direct cyanidation (over a 72 hour hour period) proved to be very successful. Extractions in excess of 94.4% were observed during these conditions. The long leaching time was required to dissolve the free gold present.

2.0 Addition of $PbNO_3$ during an eight hour pre-aeration stage produced a much lower gold extraction than without this stage (less than 74.2%).

3.0 Bottle roll cyanidation tests on 19 mm (3/4"), 12.7 mm (1/2"), 9.5 mm (3/8"), 6 mm (1/4") ore proved that at this size range, there was virtually no difference in the final gold extractions.

4.0 Column heap leaching of the TR21N sample showed an increase in rate of gold extraction for the finer size feed, but the overall extraction was basically the same up to a 38 mm (1 1/2") nominal size. The 76 mm (3") and run of mine material produced gold extractions similar to each other.

5.0 Cyanide consumptions were high for these column leach tests due to the low pH level of 10.5-11.0. A pilot test was run at a pH of 11.0-11.5, which lowered the cyanide consumption dramatically from an average of 6.66 to 2.04 kg/t. This figure of 2.04 kg/t may be improved even further if the head leach feed was mixed with lime during heap construction which is normal practice in an operating plant. Further testing of samples from the deposit is required to optimize reagent consumptions for the ore type.

6.0 The diamond drill core representing the sulphide and transition zones of the deposit (Antoniuk holes 21 and 24) showed extractions that were generally low (below 40.1%). There were two exceptions, the oxide and a mixture of transition I and II zones which produced extractions of 61.1 and 58.0% respectively. Cyanide consumptions were also high for these tests which may be improved by operating the leaching conditions with higher lime levels.

APPENDIX I

AGITATED LEACH SUMMARY SHEETS

CYANIDATION TEST SUMMARY

TEST NO. : 9

PURPOSE : standard agitated leach test

FEED : 501.1 grams of Archer-Cathro TR21N composite (-6 mesh material)

GRIND : 10 minutes; 65 % solids

LEACH : 48 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 2.31 Lime : 5.43

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	635	0.36	0.23	63.9
Wash	1025	0.05	0.05	13.9
Titration	375	<0.03	-	-
Residue	497.6	0.16 *	0.08	22.2
Total		(0.72)	0.36	100.0

Au EXTRACTION : 77.8 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.72 g/t
MASS BALANCE : 77.4 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	30.1	0.17	0.005
-150	224.35	(0.16;0.15)	0.035
total	254.45	0.16	0.045

CYANIDATION TEST SUMMARY

TEST NO. : 10

PURPOSE : duplicate test No. 9

FEED : 490.8 grams of Archer-Cathro TR21N composite
(-6 mesh material)

GRIND : 10 minutes; 65 % solids

LEACH : 48 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 2.18 Lime : 5.75

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	610	0.42	0.26	68.5
Wash	1030	0.06	0.06	15.7
Titration	350	<0.03	-	-
Residue	488.6	0.12 *	0.06	15.8
Total		(0.77)	0.38	100.0

Au EXTRACTION : 84.2 % CALC. HEAD : 0.77 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 97.5 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	21.89	0.16	0.004
-150	214.8	(0.10;0.12)	0.024
total	236.69	0.12	0.028

CYANIDATION TEST SUMMARY

TEST NO. : 11

PURPOSE : standard agitated leach

FEED : 495.7 grams of Archer-Cathro TR18N composite
(-6 mesh material)

GRIND : 10 minutes; 65 % solids

LEACH : 48 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 2.06 Lime : 6.33

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	648	2.09	1.35	74.3
Wash	1000	0.31	0.31	17.0
Titration	380	0.12	0.05	2.7
Residue	493.0	0.23 *	0.11	6.0
Total		(3.67)	1.82	100.0

Au EXTRACTION : 94.0 % CALC. HEAD : 3.67 g/t
ASSAYED HEAD : 5.10 g/t MASS BALANCE : 72.0 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	14.24	0.39	0.006
-150	235.5	(0.21;0.20)	0.051
total	249.74	0.23	0.057

CYANIDATION TEST SUMMARY

TEST NO. : 12

PURPOSE : duplicate test No. 11

FEED : 491.9 grams of Archer-Cathro TR18N composite
(-6 mesh material)

GRIND : 10 minutes; 65 % solids

LEACH : 48 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 2.01 Lime : 6.20

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	645	2.81	1.81	76.8
Wash	1040	0.38	0.40	16.9
Titration	390	0.15	0.06	2.5
Residue	484.0	0.18 *	0.09	3.8
Total		(4.80)	2.36	100.0

Au EXTRACTION : 96.2 % CALC. HEAD : 4.80 g/t
ASSAYED HEAD : 5.10 g/t MASS BALANCE : 94.1 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	12.6	0.43	0.005
-150	235.5	0.17	0.039
total	239.65	0.18	0.044

CYANIDATION TEST SUMMARY

TEST NO. : 13

PURPOSE : standard agitated leach (72 hour retention)

FEED : 496.6 grams of Archer-Cathro TR21N composite (-6 mesh material)

GRIND : 13 minutes; 65 % solids

LEACH : 72 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS :
Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 4.31 Lime : 4.73

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	610	0.76	0.46	79.3
Wash	1000	0.11	0.11	19.0
Titration	460	<0.03	-	-
Residue	495.0	<0.03 *	0.01	1.7
Total		(1.17)	0.58	100.0

Au EXTRACTION : 98.3 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 1.17 g/t
MASS BALANCE : 148.0 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	6.89	0.03	0.00
-150	240.0	(0.03;0.03)	0.007
total	246.9	0.03	0.007

CYANIDATION TEST SUMMARY

TEST NO. : 14

PURPOSE : duplicate test No. 13

FEED : 484.1 grams of Archer-Cathro TR21N composite (-6 mesh material)

GRIND : 13 minutes; 65 % solids

LEACH : 72 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REAGENTS : Grind - water
Pre-aer'n - none
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 3.22 Lime : 4.94

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	600	0.30	0.18	94.7
Wash	1005	<0.03	0.00	-
Titration	440	<0.03	0.00	-
Residue	482.0	<0.03 *	0.01	5.3
Total		(0.39)	0.19	100.0

Au EXTRACTION : >94.4 % CALC. HEAD : 0.39 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 49.4 %

COMMENTS : * - calculated from metallics assay below

mesh	wt(g)	Au(g/t)	Au(mg)
+150	6.63	<0.03	0.00
-150	225.0	(<0.03;<0.03)	0.007
total	231.63	0.03	0.007

CYANIDATION TEST SUMMARY

TEST NO. : 15

PURPOSE : standard leach with pre-aer'n and PbNO3

FEED : 497.0 grams of Archer-Cathro TR21N composite
(-6 mesh material)

GRIND : 13 minutes; 65 % solids

PRE-AER'N : 8 hours; 40 % solids

LEACH : 72 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REPULP : after pre-aer'n

REAGENTS : Grind - water
Pre-aer'n - Lime - pH -10.0
- PbNO3 - 0.12 kg/t
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 2.45 Lime : 5.29

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	645	0.29	0.19	61.3
Wash	1010	0.04	0.04	12.9
Titration	250	<0.03	0.00	-
Residue	489.7	0.17 *	0.08	25.8
Total		(0.62)	0.31	100.0

Au EXTRACTION : 74.2 % CALC. HEAD : 0.62 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 78.5 %

COMMENTS : * - average of 0.33, 0.18, <0.03 and 0.15 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : 16

PURPOSE : duplicate test No. 15

FEED : 510.0 grams of Archer-Cathro TR21N composite
(-6 mesh material)

GRIND : 13 minutes; 65 % solids

PRE-AER'N : 8 hours; 40 % solids

LEACH : 72 hours; 40 % solids

WASH : 5 x 200 mls water - twice solids weight

REPULP : after pre-aer'n

REAGENTS : Grind - water
Pre-aer'n - Lime - pH -10.0
- PbNO3 - 0.12 kg/t
Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0
- PbNO3 - 0.12 kg/t

CONSUMPTIONS (kg/t) NaCN : 2.18 Lime : 5.59 PbNO3 : 0.25

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	630	0.37	0.23	57.5
Wash	1015	0.04	0.04	10.0
Titration	260	<0.03	0.00	-
Residue	502.6	0.25 *	0.13	32.5
Total		(0.78)	0.40	100.0

Au EXTRACTION : 67.5 % CALC. HEAD : 0.78 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 98.7 %

COMMENTS : * - average of 0.23 and 0.27 g/t Au

CYANIDATION TEST SUMMARY

TEST NO. : 17

PURPOSE : bottle rolls

FEED : 994.1 grams of Archer-Cathro TR21N composite (-3/4" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.50 Lime : 3.97

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1460	0.37	0.54	40.0
Wash	1775	0.06	0.11	8.1
Titration	300	<0.03	0.00	-
Residue	907.5	0.77 *	0.70	51.9
Total		(1.36)	1.35	100.0

Au EXTRACTION : 48.1 % CALC. HEAD : 1.36 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 172.2 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	12.6	2.62	0.033
-150	233.0	0.668	0.156
total	245.6	0.77	0.189

CYANIDATION TEST SUMMARY

TEST NO. : 19

PURPOSE : bottle rolls

FEED : 1001.5 grams of Archer-Cathro TR21N composite (-3/8" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.37 Lime : 4.09

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1370	0.40	0.55	57.3
Wash	2100	0.06	0.13	13.5
Titration	320	<0.03	0.00	-
Residue	928.1	0.28 *	0.28	29.2
Total		(0.96)	0.96	100.0

Au EXTRACTION : 70.8 % CALC. HEAD : 0.96 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 121.5 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	23.72	0.34	0.008
-150	189.1	(0.27;0.27)	0.051
total	212.82	0.27	0.059

CYANIDATION TEST SUMMARY

TEST NO. : 20

PURPOSE : bottle rolls

FEED : 1008.0 grams of Archer-Cathro TR21N composite
(-1/4" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.61 Lime : 4.86

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1380	0.37	0.51	56.0
Wash	2175	0.06	0.13	14.3
Titration	300	<0.03	0.00	-
Residue	947.0	0.27 *	0.27	29.7
Total		(0.92)	0.91	100.0

Au EXTRACTION : 70.3 % CALC. HEAD : 0.92 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 116.5 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	26.58	0.34	0.009
-150	200.9	(0.27;0.26)	0.053
total	227.48	0.265	0.062

CYANIDATION TEST SUMMARY

TEST NO. : 21

PURPOSE : bottle rolls

FEED : 1005.1 grams of Archer-Cathro TR21N composite
(-3/4" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 0.85 Lime : 4.03

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1530	0.29	0.44	55.7
Wash	2040	0.04	0.08	10.1
Titration	340	<0.03	0.00	-
Residue	988.2	0.275 *	0.27	34.2
Total		(0.79)	0.79	100.0

Au EXTRACTION : 65.8 % CALC. HEAD : 0.79 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 100.0 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	36.44	0.22	0.008
-150	203.3	(0.27;0.30)	0.058
total	239.74	0.275	0.066

CYANIDATION TEST SUMMARY

TEST NO. : 22

PURPOSE : bottle rolls

FEED : 999.6 grams of Archer-Cathro TR21N composite
(-1/2" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.34 Lime : 5.13

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1550	0.27	0.42	53.8
Wash	2060	0.04	0.08	10.3
Titration	400	<0.03	0.00	-
Residue	983.6	0.28 *	0.28	35.9
Total		(0.78)	0.78	100.0

Au EXTRACTION : 64.1 % CALC. HEAD : 0.78 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 98.7 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	25.4	0.61	0.015
-150	212.4	(0.18;0.31)	0.052
total	237.8	0.28	0.067

CYANIDATION TEST SUMMARY

TEST NO. : 23

PURPOSE : bottle rolls

FEED : 996.7 grams of Archer-Cathro TR21N composite
(-3/8" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.18 Lime : 5.21

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1590	0.25	0.40	67.8
Wash	2050	<0.03	0.00	-
Titration	380	<0.03	0.00	-
Residue	978.9	0.196 *	0.19	32.2
Total		(0.61)	0.59	100.0

Au EXTRACTION : 67.8 % CALC. HEAD : 0.61 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 77.2 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	18.06	<0.03	0.001
-150	218.9	(0.25;0.13)	0.042
total	219.08	0.196	0.043

CYANIDATION TEST SUMMARY

TEST NO. : 24

PURPOSE : bottle rolls

FEED : 992.0 grams of Archer-Cathro TR21N composite
(-1/4" material)

LEACH : 72 hours; 40 % solids

WASH : 5 x 400 mls water - twice solids weight

REPULP : none

REAGENTS : Leach - NaCN - 1.00 gpl free
- Lime - pH 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 1.41 Lime : 5.31

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	1470	0.29	0.43	61.5
Wash	2010	0.04	0.08	11.4
Titration	400	<0.03	0.00	-
Residue	987.6	0.192 *	0.19	27.1
Total		(0.71)	0.70	100.0

Au EXTRACTION : 72.9 % CALC. HEAD : 0.71 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 89.9 %

COMMENTS : * - calculated from metallics assay

mesh	wt(g)	Au(g/t)	Au(mg)
+150	25.0	0.64	0.016
-150	199.1	(0.14;0.13)	0.027
total	224.1	0.192	0.043

APPENDIX II

COLUMN HEAP LEACH SUMMARY SHEETS

CYANIDATION TEST SUMMARY

TEST NO. : 5

PURPOSE : column heap leach

FEED : 12430 grams of Archer-Cathro TR21N composite (3/4" material)

LEACH : 61 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 8.94 Lime : 0.56

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	19.1	0.06	1.15	15.7
Wash	19.3	<0.03	–	–
Carbons *	107.44	36.02	3.87	52.7
Residue	12232	0.19	2.32	31.6
Total		(0.60)	7.34	100.0

Au EXTRACTION : 68.4 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.60 g/t
MASS BALANCE : 75.9 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 6

PURPOSE : column heap leach (duplicate test No. 5)

FEED : 11390 grams of Archer-Cathro TR21N composite (3/4" material)

LEACH : 61 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 11.26 Lime : 1.76

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	19.6	0.06	1.18	9.7
Wash	19.8	<0.03	-	-
Carbons *	104.35	95.54	9.97	82.0
Residue	11176	0.09	1.01	8.3
Total		(1.09)	12.16	100.0

Au EXTRACTION : 91.7 % CALC. HEAD : 1.09 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 138.0 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 7

PURPOSE : column heap leach

FEED : 26.5 kilograms of Archer-Cathro TR21N composite
(1 1/2" material)

LEACH : 69 days

WASH : wash with 33 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 6.13 Lime : 2.31

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	20.6	0.04	0.82	3.6
Wash	32.8	<0.03	-	-
Carbons *	126.35	150.93	19.07	83.2
Residue	26374	0.115	3.03	13.2
Total		(0.87)	22.92	100.0

Au EXTRACTION : 86.8 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.87 g/t
MASS BALANCE : 110.1 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 8

PURPOSE : column heap leach (duplicate test No. 7)

FEED : 24.3 kilograms of Archer-Cathro TR21N composite
(1 1/2" material)

LEACH : 69 days

WASH : wash with 32 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 7.10 Lime : 2.54

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	20.2	0.04	0.81	4.0
Wash	31.6	<0.03	-	-
Carbons *	107.03	157.71	16.88	83.1
Residue	23704	0.11	2.61	12.9
Total		(0.86)	20.3	100.0

Au EXTRACTION : 87.1 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.86 g/t
MASS BALANCE : 108.9 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 9

PURPOSE : column heap leach

FEED : 62.4 kilograms of Archer-Cathro TR21N composite
(3" material)

LEACH : 91 days

WASH : wash with 34 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 3.43 Lime : 0.90

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	20.0	0.10	2.00	5.8
Wash	34.1	<0.03	-	-
Carbons *	125.41	171.76	21.54	62.1
Residue	60800	0.183	11.13	32.1
Total		(0.56)	34.67	100.0

Au EXTRACTION : 67.9 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.56 g/t
MASS BALANCE : 70.9 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 10

PURPOSE : column heap leach (duplicate test No. 9)

FEED : 24.3 kilograms of Archer-Cathro TR21N composite
(3" material)

LEACH : 91 days

WASH : wash with 40 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 3.47 Lime : 1.47

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	19.9	0.06	1.19	3.1
Wash	39.2	<0.03	-	-
Carbons *	126.12	189.9	23.95	61.5
Residue	62700	0.22	13.79	35.4
Total		(0.62)	38.93	100.0

Au EXTRACTION : 64.6 % CALC. HEAD : 0.62 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 78.5 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 11

PURPOSE : column heap leach

FEED : 101 kilograms of Archer-Cathro TR21N composite
(run of mine material)

LEACH : 90 days

WASH : wash with 83 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 6.61 Lime : 2.22

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	71.0	0.09	6.39	12.1
Wash	82.5	<0.03	-	-
Carbons *	126.37	229.96	29.06	55.0
Residue	109000	0.172	17.35	32.9
Total		(0.52)	52.8	100.0

Au EXTRACTION : 67.1 %
ASSAYED HEAD : 0.79 g/t
CALC. HEAD : 0.52 g/t
MASS BALANCE : 65.8 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 12

PURPOSE : column heap leach (duplicate test No. 11)

FEED : 106.5 kilograms of Archer-Cathro TR21N composite
(run of mine material)

LEACH : 90 days

WASH : wash with 83 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 6.32 Lime : 2.97

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	74.0	0.12	8.88	11.0
Wash	82.7	<0.03	-	-
Carbons *	123.91	441.21	54.67	67.4
Residue	106300	0.165	17.54	21.6
Total		(0.76)	81.09	100.0

Au EXTRACTION : 78.4 % CALC. HEAD : 0.76 g/t
ASSAYED HEAD : 0.79 g/t MASS BALANCE : 96.2 %

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 13

PURPOSE : column heap leach

FEED : 47 kilograms of Archer-Cathro drill core
Ant 2-21 oxide
Ant 11-24 oxide 1; trans 1
Ant 1-21 oxide

LEACH : 82 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 4.63 Lime : 1.74

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	18.0	0.10	1.80	1.9
Wash	19.6	<0.03	-	-
Carbons *	82.35	669.7	55.15	59.2
Residue	47100	0.77	36.27	38.9
Total		(1.98)	93.22	100.0

Au EXTRACTION : 61.1 % CALC. HEAD : 1.98 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 14

PURPOSE : column heap leach

FEED : 45 kilograms of Archer-Cathro drill core
Ant 13-24 Hyp 1 Ant 16-24 Hyp 1
Ant 14-24 Hyp 1 Ant 12-24 Hyp 1
Ant 15-24 Hyp 1

LEACH : 90 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 4.01 Lime : 0.56

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	18.1	0.07	1.27	0.7
Wash	20.8	<0.03	-	-
Carbons *	82.86	653.15	54.12	27.9
Residue	45100	3.08	138.91	71.5
Total		(4.31)	194.3	100.0

Au EXTRACTION : 28.5 % CALC. HEAD : 4.31 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 15

PURPOSE : column heap leach

FEED : 45 kilograms of Archer-Cathro drill core
Ant 12-24 Trans 1 Ant 17-24 Trans 2
Ant 11-24 Trans 1 Ant 18-24 Trans 2

LEACH : 90 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 4.92 Lime : 1.51

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	17.1	0.14	2.39	0.8
Wash	19.2	<0.03	-	-
Carbons *	81.62	1980.89	161.68	57.2
Residue	37900	3.13	118.63	42.0
Total		(7.46)	282.7	100.0

Au EXTRACTION : 58.0 % CALC. HEAD : 7.46 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 16

PURPOSE : column heap leach

FEED : 36 kilograms of Archer-Cathro drill core
Ant 17-24 Hypo 1
Ant 18-24 Hypo 2

LEACH : 82 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 4.61 Lime : 1.03

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	18.5	0.06	1.11	1.9
Wash	20.8	<0.03	-	-
Carbons *	88.45	255.51	22.6	38.2
Residue	33460	1.00	35.46	59.9
Total		(1.67)	59.17	100.0

Au EXTRACTION : 40.1 % CALC. HEAD : 1.67 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 17

PURPOSE : column heap leach

FEED : 45 kilograms of Archer-Cathro drill core
Ant 7-21 Hypo 1 Ant 8-21 Hypo 1
Ant 6-21 Hypo 1 Ant 9-21 Hypo 1
Ant 10-21 Hypo 1

LEACH : 82 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t) NaCN : 3.64 Lime : 0.85

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	17.6	0.04	0.70	1.6
Wash	19.6	<0.03	-	-
Carbons *	81.21	198.62	16.13	36.8
Residue	45000	0.60	27.00	61.6
Total		(0.97)	43.83	100.0

Au EXTRACTION : 38.4 % CALC. HEAD : 0.97 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 18

PURPOSE : column heap leach

FEED : 38 kilograms of Archer-Cathro drill core
Ant 6-21 Trans 1   Ant 4-21 Trans 1
Ant 5-21 Trans 1   Ant 3-21 Trans 1

LEACH : 82 days

WASH : wash with 20 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 10.5 - 11.0

CONSUMPTIONS (kg/t)   NaCN : 5.01   Lime : 1.81

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	18.3	0.05	0.92	2.7
Wash	20.2	<0.03	-	-
Carbons *	82.83	132.68	10.99	32.8
Residue	27550	0.575	21.59	64.5
Total		(0.89)	33.5	100.0

Au EXTRACTION : 35.6 %   CALC. HEAD : 0.89 g/t

COMMENTS : * - total of all carbons

CYANIDATION TEST SUMMARY

TEST NO. : 19

PURPOSE : column heap leach

FEED : 305.5 kilograms of Archer-Cathro TR21N Zone 4
(1 1/2" material)

LEACH : 86 days

WASH : wash with 170 litres of tap water

REAGENTS : NaCN - 0.80 gpl free
Lime - pH - 11.0 - 11.5

CONSUMPTIONS (kg/t) NaCN : 2.04 Lime : 3.90

Au MASS BALANCE

Product	WT/VOL g/l	Au ppm	Au mg	Au Dist'n
Filtrate	58	<0.03	-	-
Wash	168	<0.03	-	-
Carbons *	366.79	441.91	162.09	74.1
Residue	305500	0.185	56.52	25.9
Total		(0.72)	218.61	100.0

Au EXTRACTION : 74.1 %
ASSAYED HEAD : 0.92 g/t
CALC. HEAD : 0.72 g/t
MASS BALANCE : 78.3 %

COMMENTS : * - total of all carbons

APPENDIX III

COLUMN HEAP LEACH RESIDUE ANALYSIS

Two metallic assays were performed on each heap leach residue. The following tables show the results of these assays.

Heap Leach No.	Mesh	Weight (g)	Au g/t	Au (mg)
5A	+150	2.48	1.27	0.003
	-150	326.70	(0.26;0.26)	0.085
	total	329.18	0.27	0.088
5B	+150	27.40	0.11	0.003
	-150	358.60	(0.11;0.12)	0.041
	total	386.00	0.11	0.044
6A	+150	40.10	0.03	0.001
	-150	298.00	0.16	0.048
	total	338.10	0.14	0.049
6B	+150	11.50	0.03	0.00
	-150	372.70	(0.06; 0.03)	0.017
	total	384.20	0.04	0.017
7A	+150	42.80	0.03	0.001
	-150	378.50	(0.13;0.08)	0.040
	total	421.30	0.10	0.041
7B	+150	23.20	0.92	0.021
	-150	411.90	(0.09;0.08)	0.035
	total	435.10	0.13	0.056

Heap Leach No.	Mesh	Weight (g)	Au g/t	Au (mg)
8A	+150	9.7	0.03	0.00
	-150	316.7	(0.03; 0.20)	0.036
	total	326.4	0.11	0.036
8B	+150	21.0	0.03	0.001
	-150	355.1	(0.12; 0.11)	0.041
	total	376.1	0.11	0.042
9A	+150	32.2	0.22	0.007
	-150	275.0	(0.16; 0.18)	0.048
	total	307.2	0.179	0.055
9B	+150	18.5	0.46	0.009
	-150	364.0	(0.20; 0.14)	0.062
	total	382.5	0.186	0.071
10A	+150	8.2	0.61	0.005
	-150	253.0	(0.19; 0.15)	0.043
	total	261.2	0.184	0.048
10B	+150	0.0	-	-
	-150	254.7	(0.24; 0.26)	0.064
	total	254.7	0.25	0.064
11A	+150	36.0	0.18	0.006
	-150	379.5	(0.19; 0.31)	
			(0.17; 0.27)	0.089
	total	415.5	0.23	0.095
11B	+150	19.9	0.23	0.005
	-150	301.8	(0.16; 0.05)	0.032
	total	321.7	0.115	0.037

Heap Leach No.	Mesh	Weight (g)	Au g/t	Au (mg)
12A	+150	39.9	0.19	0.008
	-150	304.7	(0.16; 0.15)	
			(0.30; 0.10)	0.047
	total	344.6	0.16	0.055
12B	+150	43.7	0.13	0.006
	-150	198.4	(0.17; 0.18)	0.035
	total	242.1	0.169	0.041
14A	+200	42.8	7.06	0.302
	-200	354.0	(1.17; 1.10)	0.402
	total	396.8	1.77	0.704
14B	+200	7.1	60.8	0.432
	-200	190.0	(1.28; 0.98)	0.215
	total	197.1	3.28	0.647
15A	+200	28.7	16.4	0.471
	-200	320.0	(1.58; 1.70)	0.525
	total	348.7	2.86	0.996
15B	+200	27.1	10.1	0.274
	-200	240.0	(1.91; 1.18)	0.371
	total	467.1	2.41	0.645

The following column heap leach residue results were obtained by shatterboxing the entire sample to 100% -150 mesh.

Heap Leach No.	Au (g/t)
13A	0.60; 1.07
13B	0.67; 0.73
14A	7.32; 1.54
14B	2.83; 1.72
15A	4.24; 2.25
15B	2.37; 4.67
16A	1.28; 1.06
16B	0.53; 1.14
17A	1.25; 0.54
17B	0.65; 0.55
18A	0.71; 0.75
18B	0.35; 0.49
19A	0.19; 0.17
19B	0.18; 0.20

Volume

4

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DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY

PHASE II

VOLUME 2
MINERALOGY AND DYE PENETRATION TESTWORK

Witteck Project 5209

for

Dept. of Supply and Services
11 Laurier Street
Phase III - II C.I.
Hull, Quebec
K1A 0S5

April, 1987

MICROMEDIA

Greg Davison
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1.0 SUMMARY

Mineralogical studies and metallurgical testwork, specifically permeability characterization, were performed on several ore samples from the Antoniuk (Mt. Freegold) gold deposit. The ore samples were collected from surface trenches TR18, TR20, TR21 and TR24. Anomalous gold values are reported in the Antoniuk Breccia and this section of the Phase II testwork was conducted to evaluate the amenability of the Antoniuk material to gold recovery by low cost heap leach cyanidation.

The dye penetration testwork indicated weak to excellent fracture controlled permeability and also, weak to extensive diffuse matrix and microfracture penetration. The degree and type of permeability was highly variable from sample to sample and/or trench to trench. The variability was controlled by the degree of silicification, sericitization and oxidation; the degree of brecciation and hydrofracturing; and the clast to matrix ratio within each ore sample. Indications that gold mineralization may be fracture controlled and the wide variability in fracture density within small samples suggest that the Antoniuk breccia is highly amenable to heap leaching of relatively coarse crushed material.

The mineralogical studies indicated that the gold principally occurred as 50 to 300 micron, liberated particles of native gold with lesser associations with iron oxyhydroxides, silicates and sulfides. Base metal sulfides are common and include sphalerite, chalcopyrite, molybdenite and galena. Supergene and hypogene oxidation have produced extensive iron oxyhydroxides after hypogene pyrite; chalcocite, covellite and native copper after chalcopyrite; and manganese oxides.

The presence of free gold, with relatively low silver content should present no problems for gold extraction. Cyanogens, including native copper and covellite, are not in sufficient quantity to inhibit leachability of the Antoniuk ore samples.The structural control of ore mineral deposition will be the prime factor governing the degree of gold recovery.

The mineralization and alteration assemblage are typical of an epithermal porphyry system. The Antoniuk Deposit probably occurs in the transitional precious metal-base metal hydrobreccia zone below the epithermal Hg-Sb-Au-Ag-SO_4 assemblage (Dart deposit) and stratigraphically above the bonanza vein deposits (La Forma-Rambler).

WD1

2.0 INTRODUCTION

The Phase I metallurgical, mineralogical and engineering assessment of the Dawson Range Heap Leach Project was conducted on several bulk samples from the Antoniuk (Mt. Freegold) Deposit.

The results indicated that the Antoniuk ore was highly amenable to gold recovery by low cost heap leach technology. The purpose of the Phase II testwork was to:

1. Assess the viability of heap leaching in the Yukon from a climatic, environmental and metallurgical standpoint.

2. Investigate fully the metallurgical characteristics of the Antoniuk (Mt. Freegold) deposit to determine if heap leaching should be considered as a possible method for gold recovery.

3. Perform a capital and operating cost estimate for a suitable sized heap leach to handle the Freegold ore.

4. Examine possible alternative methods of gold recovery such as vat leaching and tank leaching or gravity concentration.

This report covers the mineralogical studies and the dye penetration testwork of the metallurgical program.

The program was conducted on four bulk surface samples from the Antoniuk Deposit. The samples were as follows:

1. Antoniuk TR18 zone
2. Antoniuk TR20 zone
3. Antoniuk TR21 zone
4. Antoniuk TR24 zone

3.0 PROCEDURES

3.1 Phase I Heap Leach Residue

Assays from the TR24 heap leach residues reported gold values in the minus 1/2 inch fraction 3 - 5 times higher than the coarse +1/2 inch residue. Subsequent rewash tests of the - 1/2 inch material were performed after coning, quartering and crushing to -6 mesh. After gold distributions from the rewash tests exhibited similar results (low filtrate assays), mineralogical samples of approximately 1200 grams were riffled from the rewash analysis reject (-6 mesh material). Similarly, the + 1/2 inch fraction was split, coned, quartered and crushed to -6 mesh prior to riffling of 1200 gram mineralogical samples.

3.2 Phase II Feed

The TR21 bulk sample was coned twice with the Bobcat shovel and quartered. A second stage of coning and quartering to produce 1/16 of the original bulk sample was crushed to 100% -1 inch and riffled in two splits. One split was crushed to 100% -6 mesh and subsequently riffled into 500 gram charges. Four 1000 gram aliquots were retained for mineralogical analysis of ore minerals. Four 500 gram charges were individually riffled to 100 grams for preparation of polished thin sections for gangue mineralogy. Four 27 x 46 mm polished thin sections were prepared directly from the -6 mesh material (Slide # TR21-5 through TR21-8). In addition, one 27 x 46 mm polished thin section was prepared from a four inch diameter hand sample of the TR24 bulk feed.

3.3 Dye Penetration Testwork

Sixteen rock samples of approximately four inch (10 cm) diameter representing four trenches from the Antoniuk deposit were selected for permeability testwork. The samples were chosen from the following trenches: TR18, TR20, TR21 and TR24. Each rock was washed to remove loose debris and dried with paper towel. The samples were placed in shallow pans containing a 2 cm deep layer of "Zyglo", a water washable penetrant dye.

Samples were cut after various exposure durations of one, four, ten and twenty-one days using a diamond saw. The rocks were rinsed in hot water to remove excess dye, rinsed with acetone and dried prior to photography under black light.

The dye fluoresces to a pale yellow-green coloration and indicates the degree of penetration.

After photographs are taken, the samples are replaced, in the same orientation, in the dye for further dye penetration.

3.4 Vanning for Microscopy

The representative samples of -6 mesh material comprised TR24 + 1/2 inch leach residue, TR24 - 1/2 inch leach residue and four charges of TR21 feed material. Prior to any grinding, each sample was screened at 48 mesh. The oversize material was ground in a shatterbox over very short durations (less than 10 seconds) and screened at 48 mesh. The process was repeated until the entire sample passed the 48 mesh (295 micron) screen. The stage grinding (screening) process was used to minimize generation of excessive fines.

Each of the minus 48 mesh samples was vanned using a 12 inch (30 cm) diameter vanning plaque in order to produce heavy mineral concentrates for the preparation of polished ore mounts. The tails were re-vanned to enhance recovery of the fine grained heavy mineral particles. The concentrates were then dried and weighed. Tailings samples were filtered, dried and assayed for gold (Table 1). The grade of the concentrate was calculated by subtraction from the average head grades given in Table 2.

The sample weights, assays and calculated gold distributions are given in Tables 3 and 4. One 25 mm diameter polished mount was prepared for each sample.

Table 3

GOLD DISTRIBUTION

	Sample Weight (Grams)		Au (Grams per Tonne)			Distribution Van Conc	
Sample #	Head	Van Conc.	Head*	Tail	Van Conc.	Au %	Wt %
TR24 +1/2	1164.7	2.36	0.30*	0.57+	-	-	-
TR24 -1/2	1200	1.12	1.78**	1.01+	826	43.3	0.09

* Average of 0.20, 0.25, 0.29, 0.45

** Average of 1.10, 1.50, 1.51, 2.58

+ Average of two fire assays.

Table 1

VAN TAILS ASSAYS

SAMPLE NO. TR-21

Sample #	Screen Fraction (Mesh)	Weight (Grams)	Au (gpt)	Calc. Head Assay gpt
TR21-1	+150	25.36	0.89	
	-150*	198.1	0.57	0.60
			0.55	
TR21-2	+150	21.50	2.05	
	-150*	218.70	0.61	0.66
			0.44	
TR21-3	+150	4.97	0.03	
	-150*	227.5	0.61	0.59
			0.60	
TR21-4	+150	25.22	0.71	
	-150*	226.50	0.61	0.67
			0.72	

Table 2

HEAD ASSAYS

SAMPLE NO. TR-21

Split #	Screen Fraction	Wt. (Grams)	Au (gpt)	Calc. Head Au (gpt)	Au Distribution
1	+150 mesh	0.59	44.2		11.6
				0.84	
	-150 mesh*	246.1	0.84		88.4
			0.62		
				0.925	
2	+150 mesh	16.9	6.81		25.3
				1.01	
	-150 mesh*	234.1	0.69		74.7
			2.49		

* Duplicate fire assays

Table 4

GOLD DISTRIBUTION

GOLD DISTRIBUTION

	Sample Weight (Grams)		Au (Grams per Tonne)			Distribution Van Conc	
Sample #	Head	Van Conc.	Head*	Tail	Van Conc.	Au %	Wt %
TR21-1	1000	0.65	0.925	0.60	(500.6)	35.2	.065
TR21-2	1000	0.45	0.925	0.66	(589.5)	28.7	.045
TR21-3	1000	0.84	0.925	0.59	(401.80)	36.2	.084
TR21-4	1005	0.67	0.925	0.67	(383.18)	27.6	.067

* Average Assay

3.5 Microscopy Examination

The polished mounts of vanned concentrates for each sample were examined under reflected light at magnifications of 100x, 200x, 500 and 1250x, the latter using oil immersion. The polished thin sectins were examined under transmitted and reflected light at magnifications of 50 – 200x. Grains of specific interest were photomicrographed using a 35 mm camera attached to the Leitz Laborlux 12, polarizing microscope.

Numerous grains were marked by circling with a diamond tipped marker objective for subsequent scanning electron microscopy (SEM). Prior to SEM examination, each mount and slide was carbon coated under vacuum. The thickness of the carbon layer was approximately 200 Angstroms.

The grains of interest were located and examined at magnifications up to 5000x. An energy dispersive x-ray analysis provided a qualitative spot analysis (EDAX) of the targetted particle. The resolution of the instrument is on the order of one micron.

Mineral identifications were based upon a combination of optical properties using the optical microscope, secondary electron images (SEI) from the scanning electron microscope and chemical data from the EDAX-CRT display. The EDAX spectra were photographed using a Polaroid camera attachment.

3.6 X-Ray Diffraction

Diffraction analysis of the TR21 samples were performed on a Philips X-ray diffractometer equipped with an Fe radiation tube. Riffled 20 gram samples of minus six mesh material were mechanically pulverized using a shatterbox and finish ground using a porcelain mortar and pestle. The samples were mounted on glass sections (27 x 46 mm) using acetone as the dispersion agent and allowed to dry. The samples were scanned at 1° 20 per minute using a high gain-continuous readout monitor. Three scans were performed from 10° 20 to 90° 20 to assess the peaks of the contained mineral phases. For each peak, 20 angles for Fe tubes were converted into spacing measurements. The latter used to identify the probable mineral species. Where individual minerals cannot be isolated for single peaks, a short list of minerals have been tentatively identified.

4.0 RESULTS

Based on the testwork, the following results were obtained.

4.1 Dye Penetration Testwork

4.1.1 TR18 Samples 1 - 4

The four TR18 samples exhibited moderate to strong fracture penetration, though fracture density was very low, and weak to moderate development of diffuse penetration. Sample #1 (Photos 1 through 4) exhibited rapid fracture penetration and patchy moderate diffuse penetration after very short exposures. Continued exposure resulted in incipient development of diffuse exposure adjacent to fractures. Sample #2 (Photos 5 to 6) exhibited rapid diffuse penetration of the micaceous matrix of the heterolithic breccia and was accentuated by numerous microfracture surfaces. Continued exposure (Photos 7 to 8) indicated a sharp decline in the penetration rates though increases in patchy diffuse penetration were observed. Sample #3 (Photos 9 through 12) initially exhibited poor to minor diffuse and fracture controlled penetration (Photos 9 to 10) with dye penetration observed in vug porosity. Continued exposure to the dye of 10 to 21 days (Photos 11 to 12) exhibit a progressive increase in penetration, probably enhanced by the heterogeneous composition of the matrix. The fragments exhibit minor diffuse penetration while the matrix permeability was enhanced by microfractures. Samples #4 (Photos 13 through 15) exhibited excellent rapid penetration of the porous matrix of the heterolithic breccia. Locally, several fragments exhibited fracture and diffuse penetration. Continued exposure (Photo 15) exhibited progressive development of matrix penetration and patchy diffusion in the micaceous fragments.

4.1.2 TR20 Samples 1 - 4

The four samples from Trench 20 exhibited rapid and extensive fracture penetration which was enhanced by diffuse matrix penetration adjacent to fractures after continued exposure. Photos 16 and 17 exhibit the strong fracture controlled penetration in Sample #1. Diffuse penetration was significantly slower than fracture penetration. Photos 18 and 19 clearly show the enhanced penetration of microfractures and peripheral fracture walls after longer exposures. Sample #2 (Photos 20 through 23) exhibited strong fracture penetration with very weak development of peripheral diffuse penetration. The sample has a low fracture density and may be a silicified breccia or coarse breccia fragments in silicified matrix. Sample #3 (Photos 24 through 27) was similar to sample #1 in that excellent fracture penetration was observed with enhanced development of weak diffuse penetration peripheral to fracture surfaces. Sample #4 (Photos 28 through 31) was strongly fractured and dye penetration was moderate to rapid with very minor diffuse penetration. Increased penetration into the fine microfractures was observed after ten days of exposure (Photo 30) while the poor results of continued exposure may be misleading due to variable fracture density (Photo 31).

4.1.3 TR21 Samples 1 - 4

The four samples from Trench 21 exhibited weakly developed fracture controlled penetration of dye (Photos 32 through 46) and locally minor diffuse penetration adjacent to the fractures and within the porous matrix. Sample #1, shown in Photos 32 to 35, exhibited moderate to weak penetration of fractures with moderate to weak diffuse penetration of porous matrix material adjacent to fractures. Sample #2 (Photos 36 through 39) exhibited weak, though progressive, increases in patchy matrix penetration. The degree of penetration was controlled by microfracture density and matrix mica content. Sample #3 (Photos 40 through 42) exhibited excellent fracture penetration though matrix penetration was absent.

Fracture density was low and extensive development of microfractures and/or brecciation were not observed. Sample #4 (Photos 43 through 46) was similar to Sample #2 with minor matrix penetration controlled by minor discontinuous fractures. The degree of penetration exhibited progressive increases but remained patchy.

4.1.4 TR24 Samples 1 - 4

The Trench 24 samples exhibited widely variable penetration from very weak fracture controlled to pervasive fracture and matrix controlled permeability (Photos 47 through 59). Sample #1 (Photos 47 through 50) exhibited weak fracture controlled permeability and displayed minor enhanced penetration after longer exposures. The sample was identified as a silicified breccia with coarse fragments. Sample #2 (Photos 51 through 54) comprised a highly porous, matrix supported breccia. This group of samples exhibited excellent matrix penetration which progressively increased with longer exposures. Photo 54 displayed pervasive penetration of the matrix after exposure of 21 days. Sample #3 (Photo 55) exhibited moderate to strong fracture controlled penetration. The sample, which broke into numerous pieces after the initial observations, was a silicified breccia, similar to Sample #1. Sample #4 (Photos 56 through 59) exhibited excellent fracture controlled penetration. Continued exposure did not enhance penetration of the sample.

4.2 Mineralogy

The following is a detailed description of the minerals and their associations within the TR21 and TR24 zones of the Antoniuk deposit. Photomicrographs and SEM-EDXA spectra are given in Appendix II.

Leach residues from the Phase I TR24 heap leach testwork were examined to identify gold mineralization, mineral associations and textures. The examination was required to clarify and understand the significance of the assay data from the + 1/2 inch and - 1/2 inch screen fractions of the leach residue.

The Phase II heap leach feed materials were examined to identify gold particles, mineral associations and mineral chemistry. In addition, all accessory minerals from the heavy mineral vanned concentrates were studied, particularly to identify deleterious minerals such as cyanogens. Polished thin sections were examined to identify gangue mineralogy, alteration phases and textural associations of the gangue with mineralization.

4.2.1 Heap Leach Residue TR24 +1/2 Inch

The +1/2 inch leach residue was chiefly comprised of arsenopyrite and pyrite with lesser quantities of magnetite-hematite, pyrrhotite, marcasite, sphene, rutile, chalcopyrite, sphalerite, galena, tetrahedrite, covellite and zircon.

Pyrite and arsenopyrite occurred as angular, anhedral to locally euhedral particles ranging from less than 20 microns to maximum of 250 - 300 microns. The accessory sulfides including chalcopyrite, galena and pyrrhotite commonly formed subrounded inclusions in pyrite and arsenopyrite. Middling particles were also observed. Marcasite, tetrahedrite, and galena also formed free particles up to 20 microns in diameter. Fe-Ti oxides occurred as free angular particles and also as irregular oxidation selvages replacing primary sulfides. Covellite formed supergene sulfide replacements along fractures and incipient patchy replacement of chalcopyrite and tetrahedrite.

Two particles of gold were observed in the +1/2 inch screen fraction material. A single subrounded free particle, of 60 x 108 microns, was present. The gold was probably weakly argentiferous as indicated by the coloration and reflectance. The second gold particle occurred as a four micron inclusion within an angular particle of pyrite.

4.2.2 Heap Leach Residue TR24 - 1/2 inch

The -1/2 inch screen fraction comprised a similar mineralogical assemblage to the +1/2 inch material. However, the arsenopyrite:pyrite ratio exhibited a significant increase. Similar relationships were observed with a higher oxide:sulfide ratio. Pyrrhotite particles were distinctly higher in abundance while the paucity of pyrite was conspicuous. Accessory sulfides including chalcopyrite, sphalerite, galena, and covellite were observed.

Forty occurrences of gold particles were reported from the vanned concentrate. The particles formed free, middling and locked associations with Fe-Ti oxides, pyrite, arsenopyrite and galena. The particle sizes ranged from one micron to 300 microns in length and equant to elongate particles were present. The associations and particle size distributions are listed below in Table 5.

Table 5

GOLD PARTICLES -1/2 INCH FRACTION

Particle No.	Particle Size (microns)	Shape	Mineral Association
1	200 x 300	Rounded	Free
2	50 x 75	Subrounded	Free
3	12 x 16	Oblate angular	Free
4	70 x 150	Collomorphic	Free
5	12 x 22	Subrounded	Free
6	35 x 40	Rounded	Free
7	1.5 x 20	Elongate	Free
8	16 x 30	Elongate	Free
9	20 x 40	Subrounded	Free
10	18 x 25	Angular	Free
11	4 x 20	Elongate	Free
12	40 x 115	Collomorphic	Free
13	20 x 20	Irregular	Free
14	8 x 15	Irregular	Free
15	100 x 100	Subangular	Free
16	35 x 30	Subangular	Free
17	60 x 110	Subangular	Free
18	20 x 45	Subangular	Free
19	29 x 38	Equant subrounded	Middling/pyrite

Table 5, cont'd

Particle No.	Particle Size (microns)	Shape	Mineral Association
20	40 x 75	Irregular, lenticles	Middling/Fe oxides/ Fe hydroxides
21	8 x 0.2	Elongate	" "
22	40 x 60	Equant	" "
23	30 x 40	Equant	" "
24	6 x 6	Equant	" "
25	14 x 14	Equant	" "
26	1.5 x 6	Elongate	" "
27	2 x 2	Rounded	" "
28	80 x 60	Elongate	" "
29	10 x 10	Equant subrounded	" "
30	70 x 180	Equant subrounded	" "
31	6 x 8	Equant subrounded	" "
32	1 x 1.5	Equant subrounded	" "
33	1 x 4	Elongate	Locked/Fe oxides/ Fe hydroxides
34	1 x 2	Equant	" "
35	14 x 0.5	Lenticle	" "
36	6 x 16	Lenticle	" "
37	3 x 5	Equant	" "
38	20 x 30	Irregular	" "
39	3.5 x 15	Subround elongate	Locked/arsenopyrite
40	2 x 6	Elongate	Locked/arsenopyrite

4.2.3 Heap Leach Feed TR 21

The TR21 feed material was chiefly comprised of pyrite, arsenopyrite, pyrolusite, Fe-Ti oxides and oxyhydroxides with lesser sphene, rutile, chalcopyrite, sphalerite, tetrahedrite, galena, molybdenite, pyrolusite, native copper, gold-electrum, chalcocite, covellite and digenite. Gangue mineralogy was dominated by quartz, alkali-feldspar and sericite-muscovite-illite with minor calcite, ankerite/siderite, albite-oligoclase, chlorite, epidote-zoisite, barite, zircon and clay minerals, tentatively kaolinite. Photomicrographs, given in Appendix II (Photos 60 through 123), document the mineralogical character of the TR 21 material.

Gold, Silver

Fifty-four occurrences of gold mineralization were observed in four polished mounts of the TR21 feed (TR 21-1, TR 21-2, TR 21-3, TR 21-4). The number of occurrences, their particle size and mineral association are listed in Table 6. The following data (Table 7) outlines the size distribution of the gold particles.

Table 6

GOLD PARTICLES - TR 21 FEED

Particle No.	Slide No.	Particle Size (microns)	Mineral Association
1	TR 21-1	45 x 50	Free
2	TR 21-1	15 x 25	Free
3	TR 21-1	85 x 140	Free
4	TR 21-1	14 x 14	Free
5	TR 21-1	8-20 x 100	Free
6	TR 21-1	25 x 45	Free
7	TR 21-1	50 x 50	Free
8	TR 21-1	25 x 25	Free
9	TR 21-1	12 x 25	Middling/silicates
10	TR 21-1	20 x 45	Middling/silicates
11	TR 21-1	16 x 20	Middling/Fe-Ti oxides
12	TR 21-1	20-30 x 120	Middling/silicates
13	TR 21-2	30 x 90	Free
14	TR 21-2	15 x 15	Free
15	TR 21-2	18 x 18	Free
16	TR 21-2	8 x 15	Middling/silicates
17	TR 21-2	16 x 40	Middling/silicates
18	TR 21-2	30 x 90	Middling/Fe-Ti oxide
19	TR 21-3	170 x 250	Free
20	TR 21-3	40 x 64	Free
21	TR 21-3	16 x 180	Free
22	TR 21-3	20 x 50	Free
23	TR 21-3	140 x 100	Free
24	TR 21-3	35 x 50	Free
25	TR 21-3	130 x 150	Free
26	TR 21-3	25 x 40	Free
27	TR 21-3	130 x 240	Free
28	TR 21-3	25 x 30	Free
29	TR 21-3	120 x 140	Free
30	TR 21-3	110 x 240	Free
31	TR 21-3	100 x 180	Free
32	TR 21-3	70 x 100	Free
33	TR 21-3	60 x 150	Free
34	TR 21-4	8 x 30	Free
35	TR 21-4	6 x 20	Free
36	TR 21-4	10 x 30	Free
37	TR 21-4	25 x 100	Free
38	TR 21-4	14 x 32	Free
39	TR 21-4	54 x 122	Free
40	TR 21-4	60-120 x 330	Free
41	TR 21-4	14 x 32	Free

Table 6, cont'd

Particle No.	Slide No.	Particle Size (microns)	Mineral Association
42	TR 21-4	2-16 x 60	Free
43	TR 21-4	25 x 35	Free
44	TR 21-4	12 x 25	Free
45	TR 21-4	30 x 60	Middling/pyrite
46	TR 21-4	12-28 x 60	Middling/pyrite
47	TR 21-4	8-30 x 90	Middling/pyrite
48	TR 21-4	75 x 82	Middling/Fe oxyhydroxide
49	TR 21-4	28 x 50	Middling/Fe oxyhydroxide
50	TR 21-4	120 x 180	Middling/Fe oxyhydroxide
51	TR 21-4	20 x 45	Locked/Fe oxyhydroxide
52	TR 21-4	4 x 7	Locked/Fe oxyhydroxide
53	TR 21-4	75 x 120	Locked/Fe oxyhydroxide
54	TR 21-4	26 x 26	Locked/Fe oxyhydroxide

Table 7

GOLD PARTICLE SIZE DISTRIBUTION

Number of Particles Long Axis	Number of Particles Short Axis	Size Range
8	19	Less than 20 microns
18	18	20-50 microns
11	8	50 - 100 microns
13	9	100 - 200 microns
4	0	Greater than 200 microns

Elongation of Particles	Degree of Elongation
31/54	Less than 2:1
17/54	Greater than 2:1, less than 4:1
6/54	Greater than 4:1

Thirty-seven of the gold particles reported as free grains ranging from equant to elongate (maximum approximately 5:1 length/width) shapes. Particle boundaries range from linear euhedral to highly irregular lenticles (Photos 60, 61, 63 and 64). The gold particles exhibit variable degree of internal porosity from nil (Photo 64) to highly porous spongy grains comprised of numerous lenticles (Photo 65). Some examples of overgrowths may be recognized where irregular fine lenticles surround coarse anhedral to subhedral gold particles (Photo 60). Scanning electron microscopy and energy dispersive x-ray analysis of the gold particles indicated that the compositions vary in silver content from trace to several percent. The range is qualitatively based upon EDXA spectra peak levels and variable reflectance. SEM-EDXA spectra of argentiferous gold (Photo 62) and high fineness gold (Photo 75) illustrate the variation in mineral chemistry. Spectra of argentiferous gold-electrum are not shown.

Middling particles accounted for only thirteen of the fifty-four occurrences. The gold typically occurs in association with Fe-Ti oxyhydroxides which replace the primary sulfide assemblage (Photos 66, 67) and with silicate gangue minerals. An irregularly shaped grain of argentiferous gold (electrum) occurred as a middling particle with a collomorphic aggregate of native copper (Photo 68). The copper particle was attached to several minute gold particles and was also adjoined by or replaced chalcopyrite. Secondary iron oxyhydroxides locally rimmed the native copper particle. SEM-EDXA spectra were used to confirm the mineral chemistry of the copper (Photo 69).

Gold mineralization was also observed as encapsulated particles with pyrite (Photos 70, 71), arsenopyrite and secondary iron oxyhydroxides (Photos 72, 73, 74). The encapsulated particles typically range in size up to a maximum of 15 microns. The inclusions form subrounded to angular shapes and range in composition from electrum to high fineness gold (Photo 75).

Iron Sulfides

Pyrite and arsenopyrite formed the main sulfide assemblage and typically occurred as coarse multi-granular aggregates of subhedral to anhedral grains. The sulfides ranged in size from less than ten microns to a maximum of 500 microns though most particles average between 125-175 microns. Examination of the TR24 polished thin section suggested that the majority of the sulfide mineralization occurred as fracture and dilation zone open space filling mineralization (Photos 76, 77, 78, and 79). Particle sizes exhibited marked variability. Iron oxyhydroxides locally formed peripheral replacement of the pyrite and, to a lesser degree, the arsenopyrite. The replacement of sulfides in the mineralized fractures ranged from trace (Photo 78) to pervasive (Photo 79). The oxyhydroxides also formed cements within the relict sulfide assemblage (Photo 77, 79). Replacement of the sulfides occurred as fracture and grain boundary controlled oxidation (Photo 80, 81) and commonly proceeded as lobate replacement fronts. Locally sulfides were retained as pseudomorphic relicts (Photo 82) or atoll structures. The preferential oxidation of pyrite relative to arsenopyrite was observed and is evidenced by the well preserved arsenopyrite euhedra (Photo 83) and the marginally replaced pyrite (also Photo 83).

Pyrrhotite formed angular free particles up to 200 microns and also occurred as subangular to subrounded inclusions in pyrite and arsenopyrite. The latter two minerals also contained inclusions, commonly less than 20 microns in diameter, of chalcopyrite, galena, sphalerite and rutile.

Copper

Copper minerals were comprised of primary tetrahedrite, and chalcopyrite with supergene development of covellite, chalcocite, digenite and native copper. The chalcopyrite occurred as angular free particles up to 175 microns and as irregular inclusions up to 75 microns in length within pyrite (Photo 87). The chalcopyrite inclusions were commonly in association with sphalerite, as exsolution blebs and lamellar intergrowths (photo 87). Chalcopyrite also formed cements between aggregates of euhedral to subhedral pyrite crystals (Photo 84). Both tetrahedrite and chalcopyrite formed free particles which exhibited peripheral supergene sulfide/oxide replacement by covellite, digenite, chalcocite and native copper (Photos 85, 86). The chalcopyrite locally contained angular to subrounded inclusions of galena, pyrite and pyrrhotite.

Base Metals

Sphalerite formed angular to subrounded free particles up to 275 microns in diameter and also occurred as angular to subrounded inclusions within pyrite. SEM-EDXA spectra of sphalerite is given in Photo 88. The sphalerite commonly contained extensive arrays of cleavage controlled chalcopyrite inclusions (Photo 89) which display a complex SEM-EDXA spectra (Photo 90). The inclusions occur as minute specks, blebs and dendritic to skeletal linear particles at intersecting cleavage surfaces of the host sphalerite.

Galena occurred as coarse angular free particles up to 250 microns in diameter (Photo 91) and as irregularly shaped encapsulated grains within pyrite and arsenopyrite (Photo 92). The inclusions range in size up to 45 microns. A single occurrence of galena and gold forming a bimineralic inclusion within pyrite was observed. The SEM-EDXA spectra of the galena (non-argentiferous) is given in Photo 93.

Molybdenite was observed as fine 10 to 20 micron platy laths. A free particle of molybdenite is shown in Photo 94 and the mineral chemistry (SEM-EDXA spectra) is shown in Photo 94. Fine whitish coloured inclusions found within arsenopyrite have been tentatively identified as tellurobismuthite (Photo 96). Several grains of tellurobismuthite were identified and qualitatively analyzed in the Phase I report.

Manganese

Tabular to sheaf-like radiating aggregates of pyrolusite, locally coated by Fe-Ti-Mn oxyhydroxides, were observed (Photos 97, 98). The grain aggregates ranged in size up to 200 microns in diameter. The SEM-EDXA spectra of the manganese oxide is given in Photo 99.

Gangue Minerals

The gangue assemblage was comprised of quartz, alkali feldspar (orthoclase/adularia), sericite-illite-muscovite, barite (Photo 100), zircon (Photo 101), calcite, siderite/ankerite, albite, chlorite, and clay minerals.

Anhedral polygonized grains of quartz intergrown with albitic to alkali (orthoclase/adularia) feldspar form the predominant mineral phases. The feldspars locally exhibit turbid kaolinization. The quartsofeldspathic intergrowths are strained with undulatory extinction and abundant microfractures. Single crystals reach 1.1 mm in diameter but typically form smaller particles of 150-300 microns occurring in multigranular aggregates.

Microfractures in the quartzofeldspathic aggregates are commonly filled by finely divided muscovite-sericite-illite (XRD identification). The white mica group of minerals also occur as isolated patches within quartz and, to a larger degree, as local replacement of alkali feldspars. Microlithons of quartz commonly occur within veinlets/fractures/dilution zones filled by white micas.

A polygonized quartz aggregate cut by a muscovite-sericite-illite veinlet is given in Photo 102. The higher birefringence particles were identified by optical and x-ray analysis as potash alkali feldspars (Photo 103).

The microfractures are commonly filled by Fe hydroxides (Photo 104) and epidote-zoisite (Photo 105) with accessory muscovite-sericite-illite (Photo 121). Offsets of microfractures were pervasive. Excellent examples of microfracture filling hy hydroxides are shown in Photo 120 and 122. The iron hydroxides commonly produced a brown stain on the white micas, feldspars and locally on the chlorite. The microfractures containing microlithons or fragments of quartzose material on host rock particles (Photo 122, 123) exhiit the discolouration of the white mica group. Lithons of quartz, shown in Photo 112, also appear to be floating in a later matrix of micaceous replacement and filling. The micaceous alteration may be observed as anastomosing veinlets with poorly developed microlithons (Photo 113). SEM-EDXA spectra of the Fe stained white mica group are given in Photo 114.

Chloritization, as Fe chlorite and Fe-Mg chlorite was prevalent in many fragments. The particles typically comprise radial to irregularly distributed sheaves of chlorite with local islands of quartzose material (Photo 106). SEM-EDXA spectra of the magnesian chlorite is given in Photo 107. Chlorite-sericite intergrowths are common and are locally discoloured by iron hydroxides (Photo 108). Similar aggregates, shown in Photos 109 and 110, exhibit the pseudomorphic replacement and degenerative oxidation of the pyritic mineralization. The chlorite-sericite alteration spectra (SEM -EDXA) are given in Photo 111.

Relict textures of the precursor volcanogenics include kaolinized feldspar phenocrysts (Photo 115), sericitized and oxidized (Fe hydroxides) phenocrysts of feldspar (Photos 117 and 118), myrmekitic intergrowths of quartz and sericite (after early feldspars) (Photo 116) and twinned plagioclase feldspars replaced by Fe hydroxides and sericite (Photo 119).

4.2.4 Heap Leach Feed TR21 - X-Ray Diffraction

The use of x-ray diffraction in the identification of fine grained replacement minerals in epithermal gold deposits is especially relevant. Recognition of the alteration mineralogy is critical to an understanding of mineral and ore body zoning. The TR21 x-ray diffraction data reports the presence of low temperature alpha quartz as the primary alteration phase. Due to peak overlap between groups of minerals, tentative identifications of the replacement assemblage was completed. A number of the following minerals were corroborated by optical and SEM-EDXA data. The mineral assemblage may include: quartz, illite, muscovite-sericite, clinochlore, microcline, adularia, dolomite, alunite, goethite, rutile and ilmenorutile. Detailed D spacing data are given in Table 8 and the diffraction profile is given in Appendix III. Data comparison indicated that quartz-muscovite-sericite-illite-chlorite-pyrite-Fe oxyhydroxides comprise the majority of the alteration assemblage.

Table 8

X-RAY DIFFRACTION ANALYSES

20 Angle	D Spacing	Mineral
25.1	4.45	Illite
26.6	4.21	Muscovite-sericite
		Goethite
		Adularia
33.7	3.34	Silica - low quartz
		Illite
		Muscovite-sericite
34.9	3.23	Rutile
		Ilmenorutile
		Microcline
39.1	2.89	Alunite
		Dolomite
		Microcline
44.7	2.54	Clinochlore
		Muscovite-sericite
		Illite
46.6	2.44	Silica - low quartz
		Goethite
50.5	2.27	Alunite
		Dolomite
		Silica - low quartz
54.3	2.12	Silica - low quartz
57.3	2.02	Clinochlore
58.7	1.98	Illite
		Muscovite-sericite
64.5	1.81	Silica - low quartz
71.0	1.67	Silica - low quartz
		Rutile
		Ilmenorutile
78.0	1.54	Clinochlore
		Silica - low quartz
89.7	1.37	Silica - low quartz

5.0 DISCUSSION

5.1 Dye Penetration Testwork

Trench samples from the Antoniuk (Mt. Freegold) deposit exhibited marked variability in degree and type of permeability. Strong variations were observed in individual samples and from different trenches.

The TR 18 material exhibited moderate to strong matrix penetration in all samples and displayed excellent heap leach potential of coarsely crushed material. In contrast to the TR18 matrix permeability, the TR20 samples display widely spaced to narrowly spaced fractures which controlled the dye penetration. Matrix penetration ranged from very weak to nil. Samples from TR21 and TR24 generally exhibited macrofracture and microfracture permeability which, in few samples, controlled local matrix penetration adjacent to the fracture surfaces. In sharp contrast, sample Number 3 from Zone TR24 exhibited excellent matrix permeability and weak to moderate penetration of fractured fragments.

The permeability contrasts are controlled by size of fragments (based on these small samples), abundance of fragments (fragment: matrix ratio), degree of sericitization, degree of silicification, degree of supergene oxidation of primary epithermal sulfides, and fracture density and dilation. Amenability to heap leaching ranges from poor to excellent based on the dye penetration results. However, indications that gold mineralization may be fracture controlled and the wide variability in fracture density within small samples suggest that the Antoniuk breccia is highly amenable to heap leaching of relatively coarse crushed material.

5.2 Mineralogy/Geology

The Antoniuk gold deposit comprises a heterolithic breccia body transected by rhyolitic porophyry dykes and numerous small scale structural dislocations. The breccia zone, part of the Mount Nansen acid to intermediate volcanics, exhibit variable degrees of fracture density due to faults, shears and secondary infilling by replacement minerals. The breccia, enclosed by granodioritic and syenitic intrusions, may comprise the upper zone of an epithermal-porphyry intrusive system.

5.2.1 Mineralization

The mineralization of the Antoniuk breccia is chiefly comprised of slightly argentiferous gold which primarily occurred as coarse free gold particles. The gold ranges in composition from high fineness gold to electrum and locally exhibited reprecipitation phenomenon of gold over electrum. The gold is locally associated as inclusions within secondary iron-oxyhydroxides after the sulfide mineralization chiefly comprised of pyrite, arsenopyrite and lesser pyrrhotite. Many additional minerals characteristic of epithermal-porphyry systems were observed and these include the chalcopyrite, sphalerite, molybdenite, tetrahedrite and tellurobismuthite. Supergene oxidation has resulted in the generation of extensive porous iron oxyhydroxides, manganese oxides, native copper, covellite, chalcocite, and digenite. However, manganese halos and zinc-silver enriched breccias are also associated with epithermal assemblages.

5.2.2 Alteration Zones

The optical microscopy and x-ray diffraction patterns indicated an assemblage of quartz-sericite-illite-muscovite + adularia + clay which characterize phyllic to silicic alteration zones of porphyry and epithermal systems. The abundance of chlorite may indicate relatively narrow zones of silicification, sericitization, and sulfidation enveloped by a propylitic halo.

The absence of antimony, mercury and acid-sulphate species, characteristic of high-level or shallow mineralization in the epithermal deposits provides depth indications. This assemblage is located at the nearby Dart mineralized zone, although some indications of stibnite, alunite and kaolinite are observed at the Antoniuk deposit. The alteration assemblages exhibit strong fracture control, probably the result of hydrobrecciation during and possibly after silicification.

6.0 CONCLUSIONS

1. Permeability of the Antoniuk samples ranged from weak fracture control to strong diffuse matrix penetration by the fluorescent dye testwork.

2. The degree of penetration exhibited wide variations from different trenches and also, from individual samples.

3. Permeability is apparently controlled by macrofracture density and size, degree of silicification, sericitization and brecciation; and clast:matrix ratios.

4. TR18 material exhiited excellent heap leach potential for coarse crushed material as did several other samples, for example TR24 #3. TR20, TR21 and TR24 samples exhibited moderate to excellent potential provided that gold mineralization is confined to macrofractures and peripheral alteration/replacement selvages.

6. The mineralogical studies of the Antoniuk trench samples reported the presence of abundant, coarse (+ 100 micron) particles of free gold. Gold also occurs as middling particles with Fe-Ti oxides/oxyhydroxides, native copper, and silicate gangue. Several gold particles were encapsulated within Fe-Ti oxyhydroxides or sulfides including pyrite and arsenopyrite.

7. Low mass balances from the vanning procedures suggest that fine-grained particles of gold were lost to tailings in association with silicate gangue and/or Fe-Ti oxides/oxyhydroxides.

8. The sulfide assemblage was comprised of pyrite and arsenopyrite with lesser to trace amounts of pyrrhotite, chalcopyrite, sphalerite, galena, tetrahedrite, molybdenite and tellurobismuthite. The supergene oxides comprised Fe-Ti oxides/ hydroxides and Mn oxides, principally pyrolusite.

9. The alteration assemblage of gangue silicate minerals comprised quartz, muscovite-sericite-illite, and chlorite. The minerals are characteristic of silicic (potassic) and phyllic-propylitic (low temperature) zones of hydrothermal alteration.

10. The mineralization and alteration zones are characteristic of an epithermal-porphyry system. The Antoniuk Deposit characterizes the hydrobreccia type occurring below the upper Hg-Sb-Mn fumarolic zone and itself contained by the regional low temperature propylitic envelope.

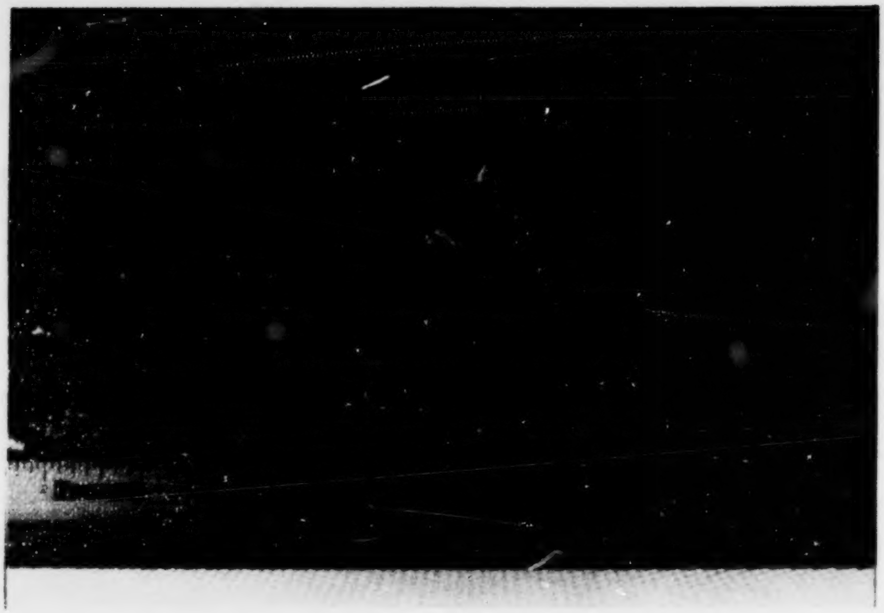
11. The presence of coarse free gold, ranging from high fineness to slightly argentiferous varieties, indicates that the ore should be highly amenable to cyanidation. Cyanogens, including native copper and covellite, are not in sufficient quantity to inhibit leachability of the Antoniuk ore samples.

APPENDIX I

Dye Penetration Testwork

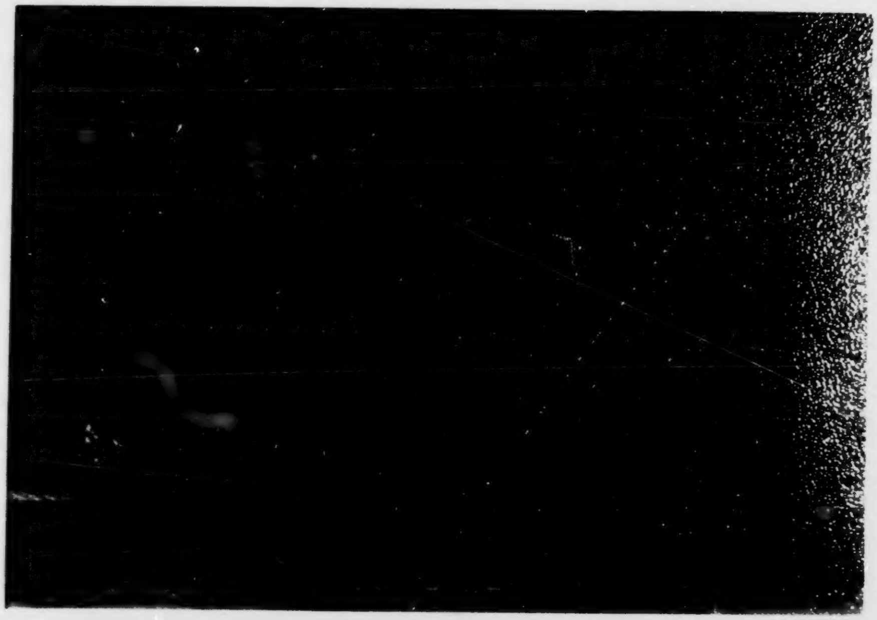
Photograph 1
TR18 Sample 1 - Exposure 1 day

Photograph 2
TR18 Sample 1 - Exposure 4 days

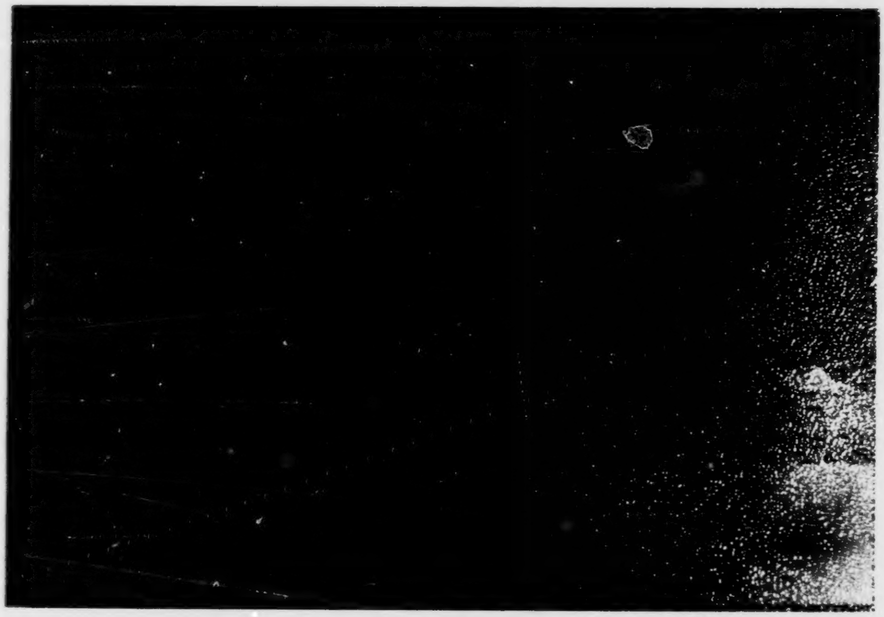


Photograph 3
TR18 Sample 1 - Exposure 10 days

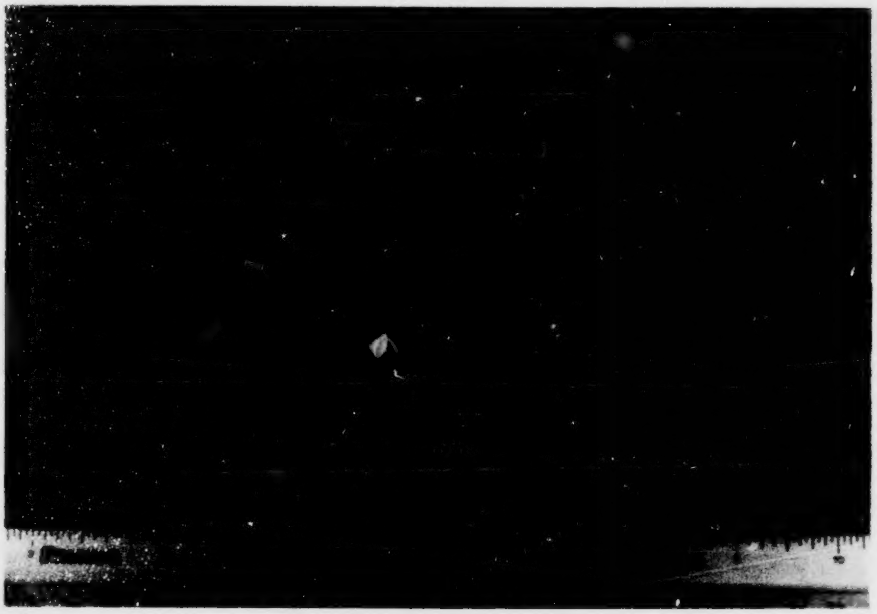
Photograph 4
TR18 Sample 1 - Exposure 21 days



Photograph 5
TR18 Sample 2 - Exposure 1 day

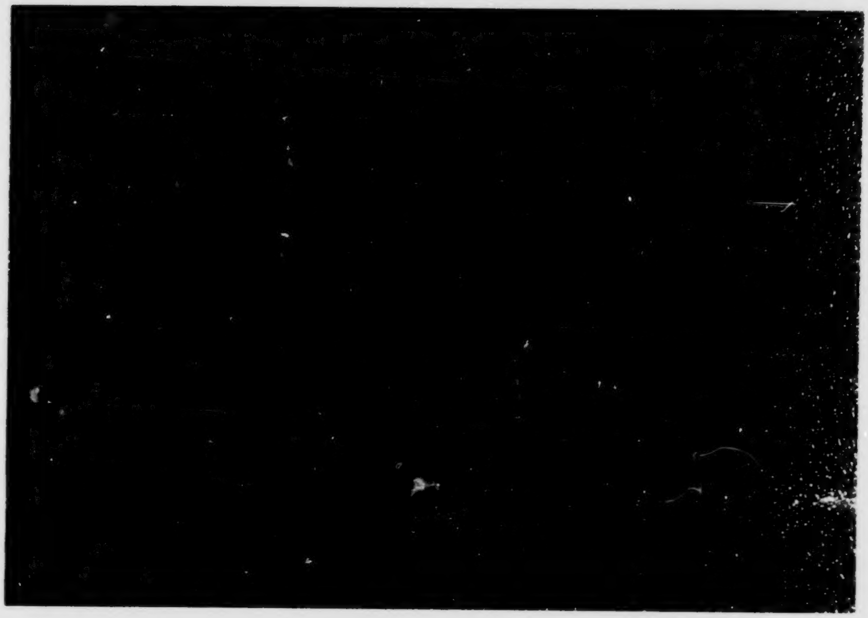


Photograph 6
TR18 Sample 2 - Exposure 4 days



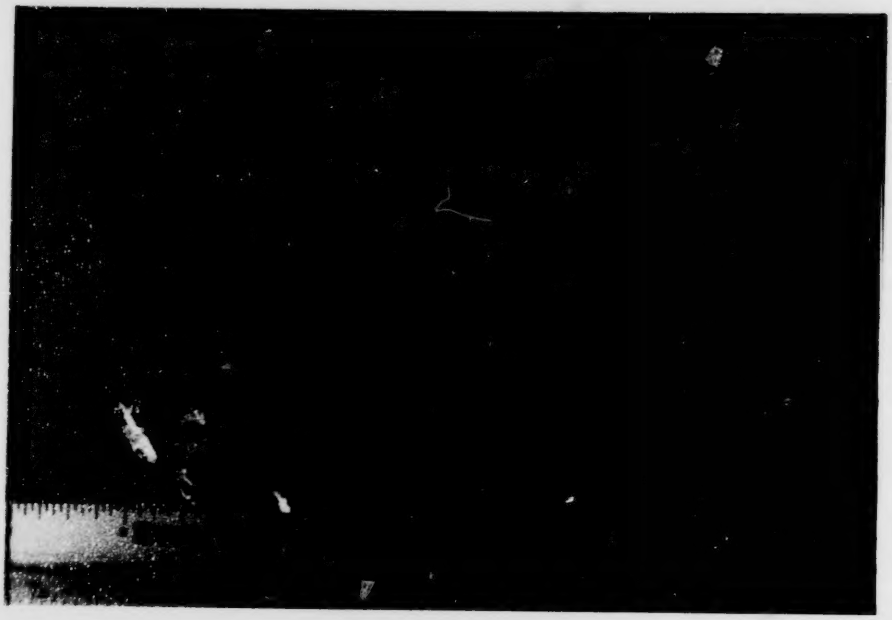
Photograph 7
TR18 Sample 2 - Exposure 10 days

Photograph 8
TR18 Sample 2 - Exposure 21 days

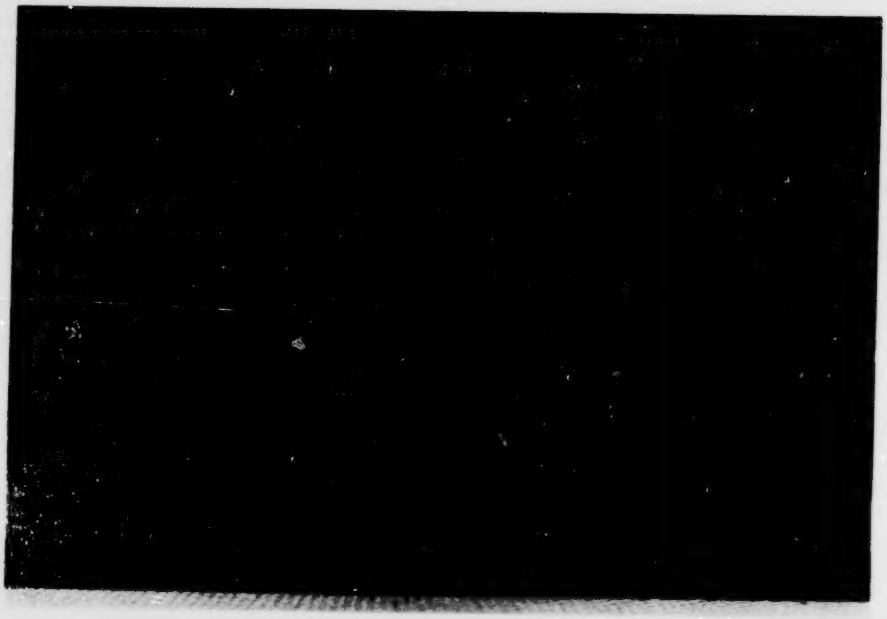


Photograph 9
TR18 Sample 3 - Exposure 1 day

Photograph 10
TR18 Sample 3 - Exposure 4 days

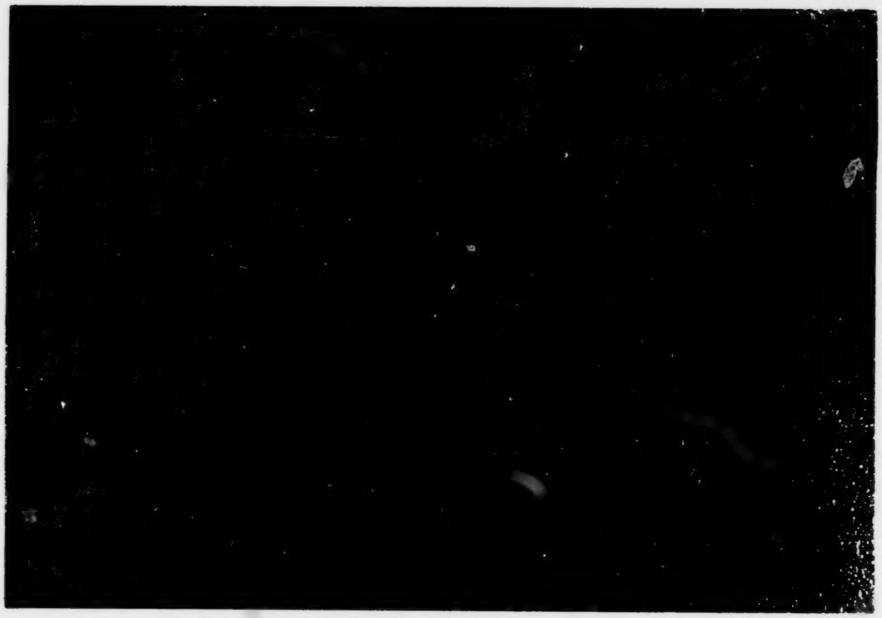


Photograph 11
TR18 Sample 3 - Exposure 10 days



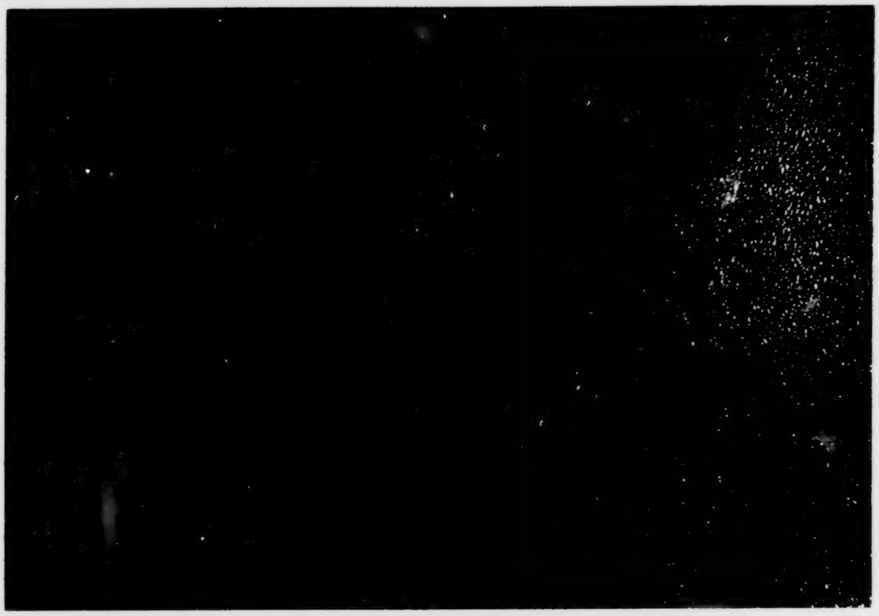
Photograph 12
TR18 Sample 3 - Exposure 21 days

Photograph 13
TR18 Sample 4 - Exposure 1 day

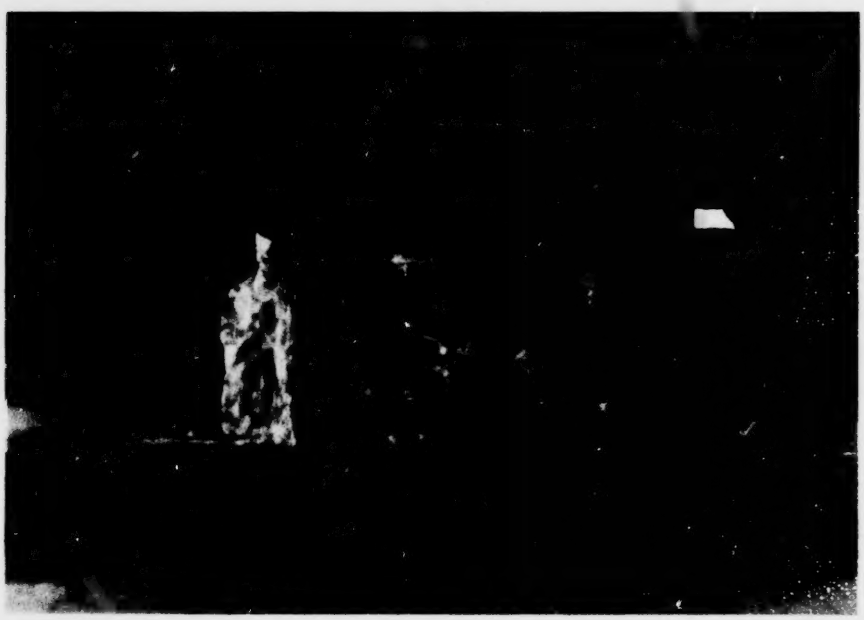


Photograph 14
TR18 Sample 4 - Exposure 4 days

Photograph 15
TR18 Sample 4 - Exposure 10 days



Photograph 16
TR20 Sample 1 - Exposure 1 day



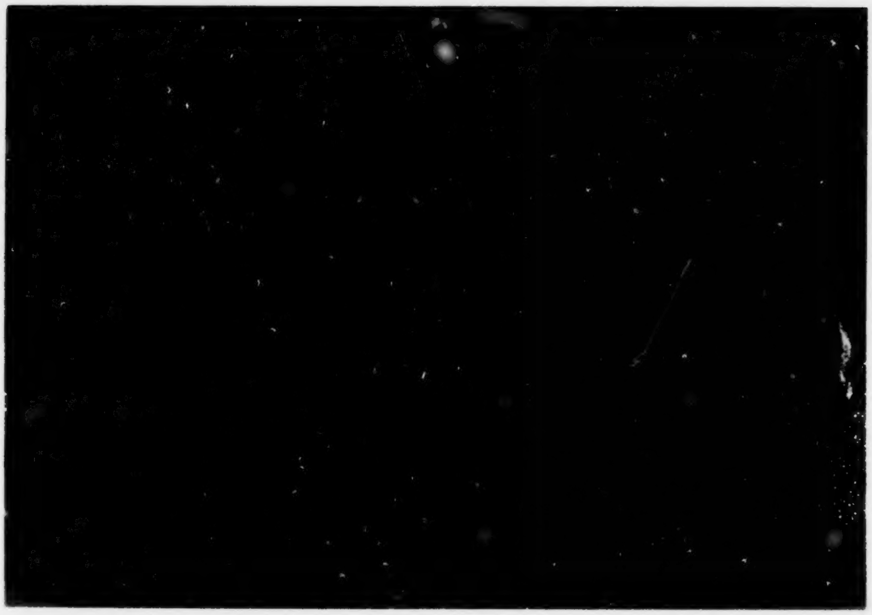
Photograph 17
TR20 Sample 1 - Exposure 4 days

Photograph 18
TR20 Sample 1 - Exposure 10 days

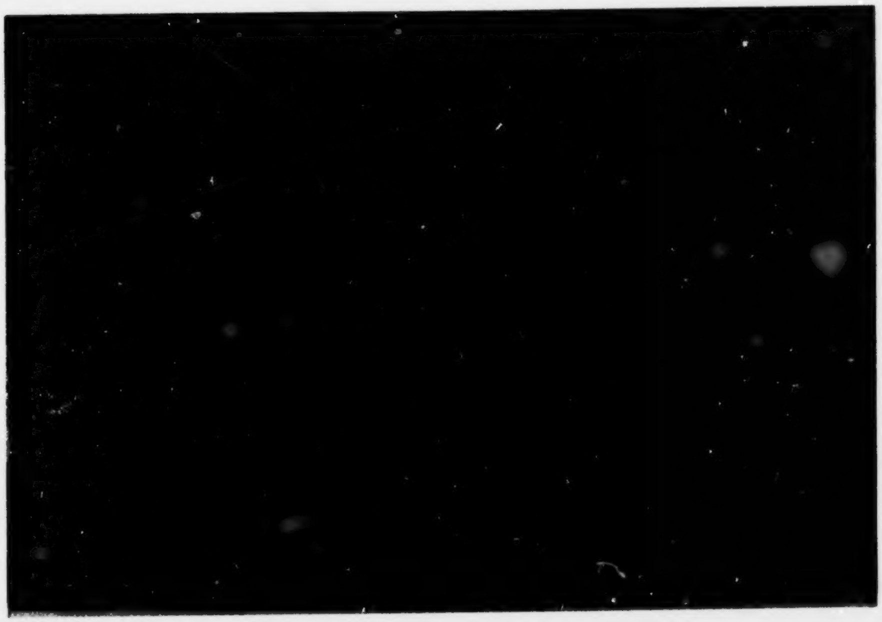
Photograph 19
TR20 Sample 1 - Exposure 21 days

WD1

Photograph 20
TR20 Sample 2 - Exposure 1 day

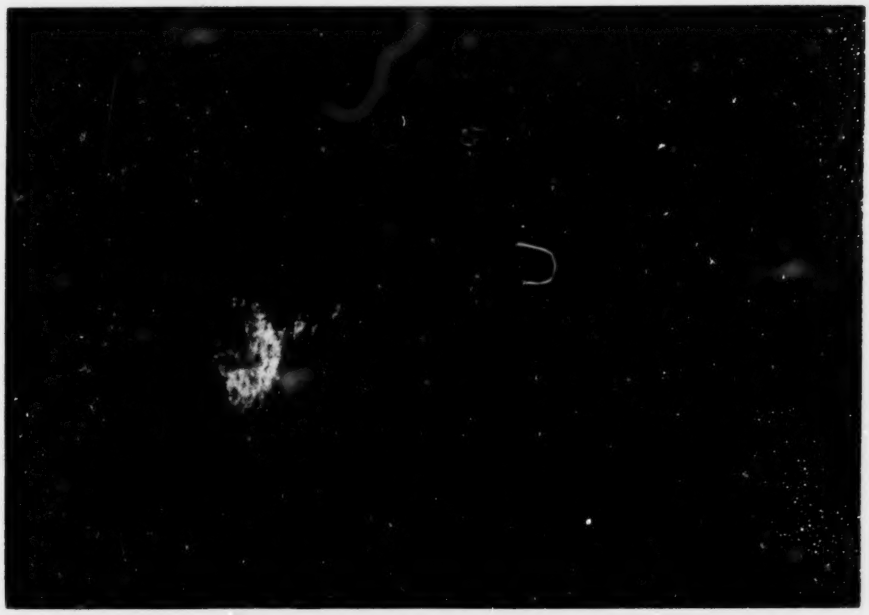


Photograph 21
TR20 Sample 2 - Exposure 4 days

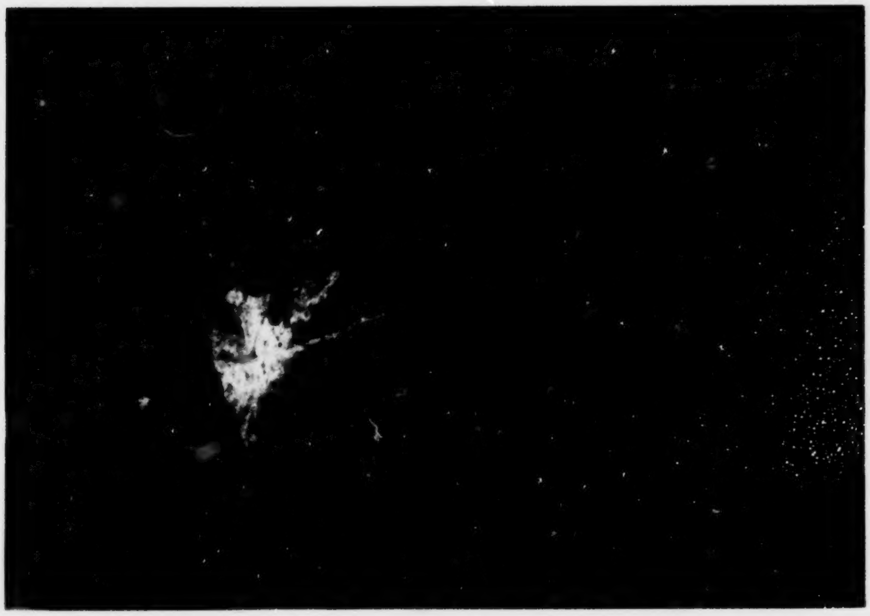


Photograph 22
TR20 Sample 2 - Exposure 10 days

Photograph 23
TR20 Sample 2 - Exposure 21 days



Photograph 24
TR20 Sample 3 - Exposure 1 day



Photograph 25
TR20 Sample 3 - Exposure 4 days

Photograph 26
TR20 Sample 3 - Exposure 10 days

Photograph 27
TR20 Sample 3 - Exposure 21 days

Photograph 28
TR20 Sample 4 - Exposure 1 day

Photograph 29
TR20 Sample 4 - Exposure 4 days

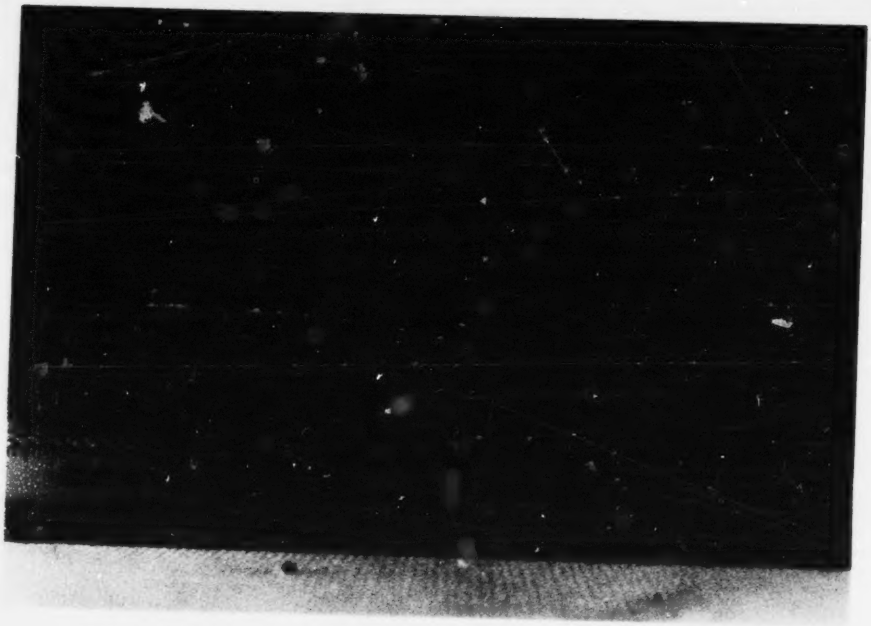
Photograph 30
TR20 Sample 4 - Exposure 10 days

Photograph 31
TR20 Sample 4 - Exposure 21 days

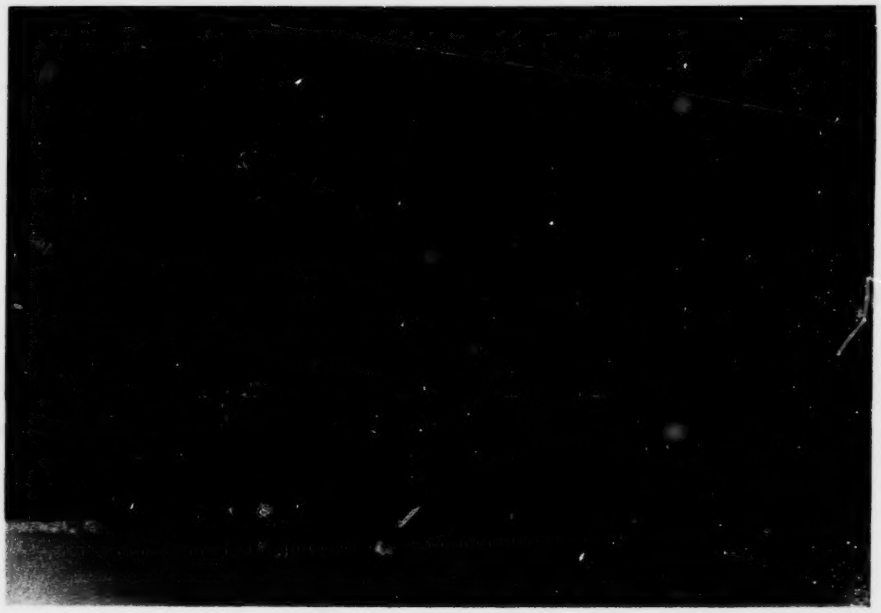
Photograph 32
TR21 Sample 1 - Exposure 1 day

Photograph 33
TR21 Sample 1 - Exposure 4 days

Photograph 34
TR21 Sample 1 - Exposure 10 days

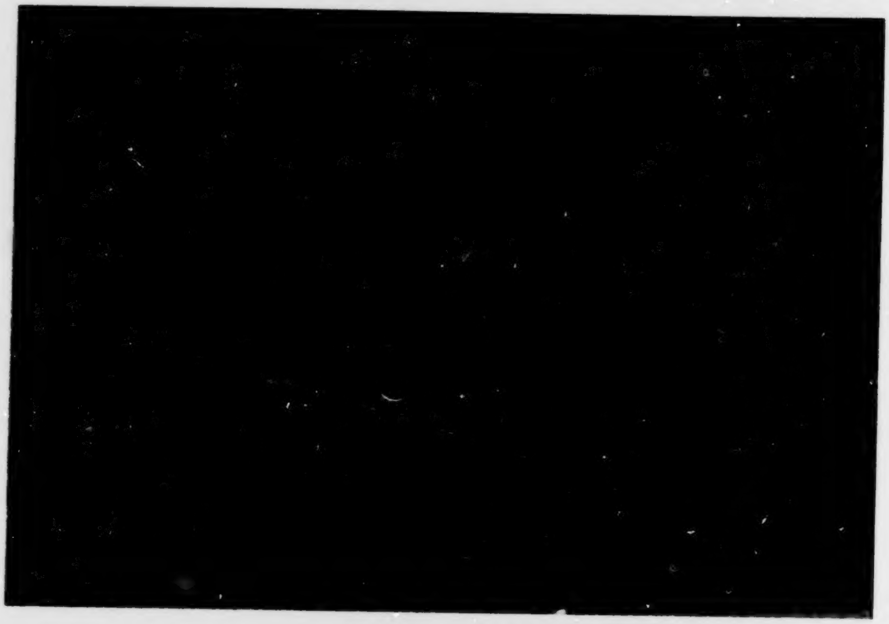


Photograph 35
TR21 Sample 1 - Exposure 21 days



Photograph 36
TR21 Sample 2 - Exposure 1 day

Photograph 37
TR21 Sample 2 - Exposure 4 days



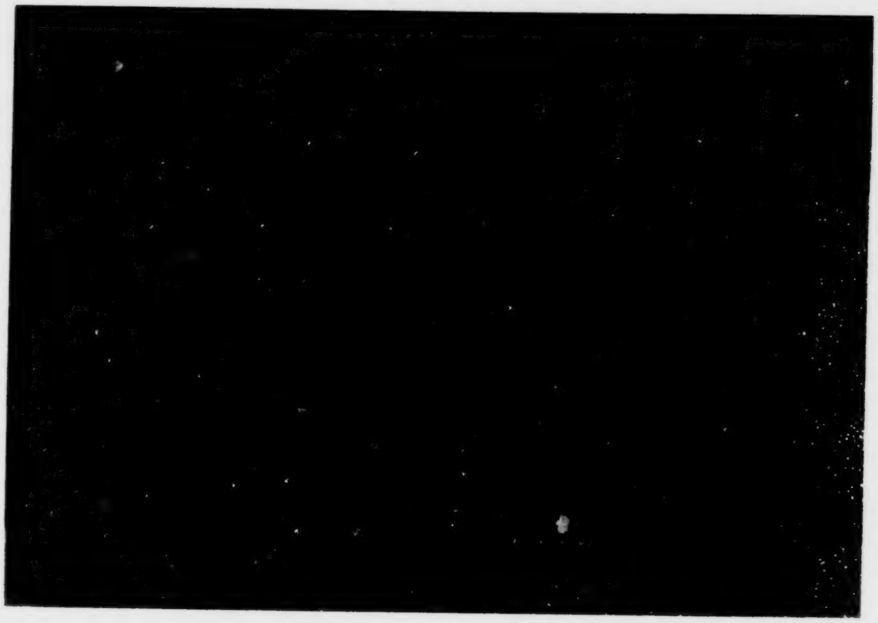
Photograph 38
TR21 Sample 2 - Exposure 10 days

Photograph 39
TR21 Sample 2 - Exposure 21 days

Photograph 40
TR21 Sample 3 - Exposure 1 day

Photograph 41
TR21 Sample 3 - Exposure 4 days

Photograph 42
TR21 Sample 3 - Exposure 10 days

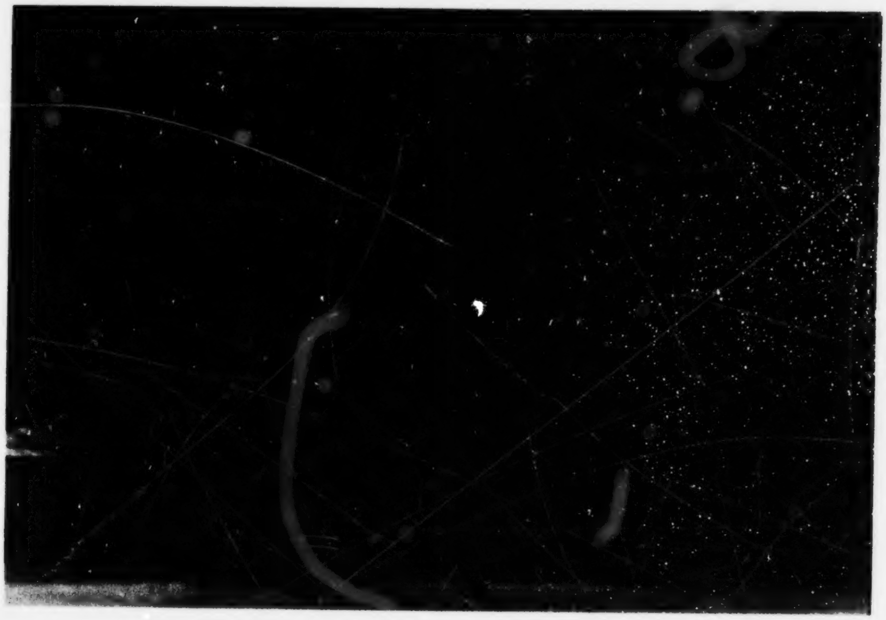


Photograph 43
TR21 Sample 4 - Exposure 1 day

Photograph 44
TR21 Sample 4 - Exposure 4 days

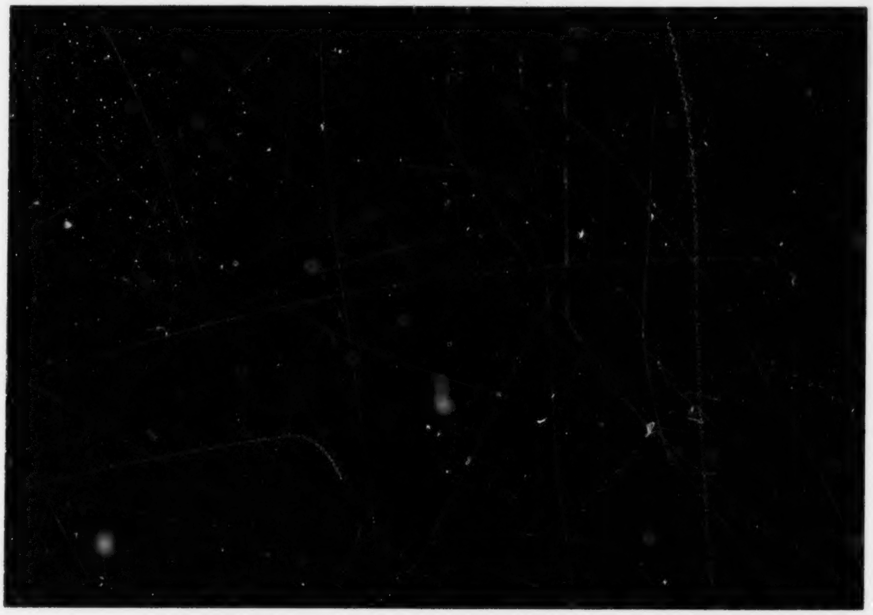
Photograph 45
TR21 Sample 4 - Exposure 10 days

Photograph 46
TR21 Sample 4 - Exposure 21 days



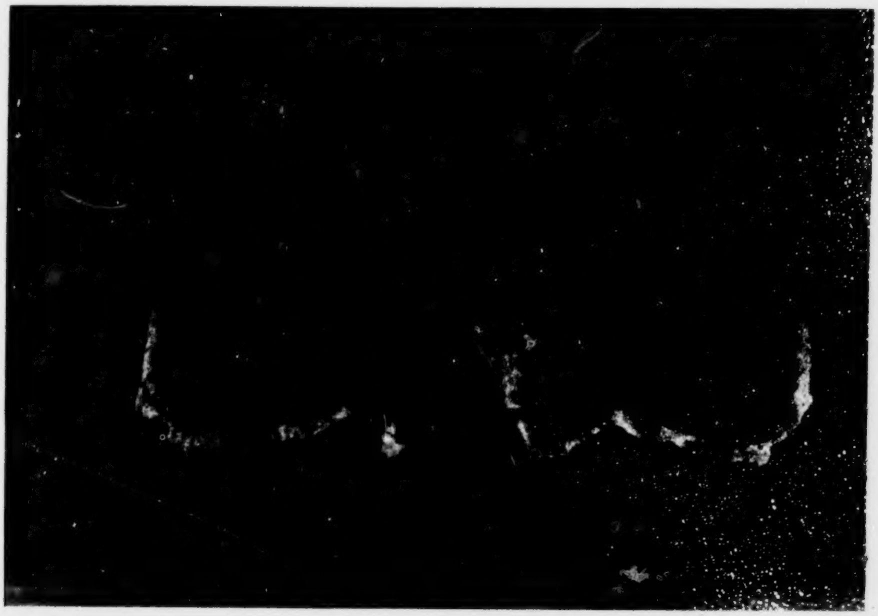
Photograph 47
TR24 Sample 1 - Exposure 1 day

Photograph 48
TR24 Sample 1 - Exposure 4 days

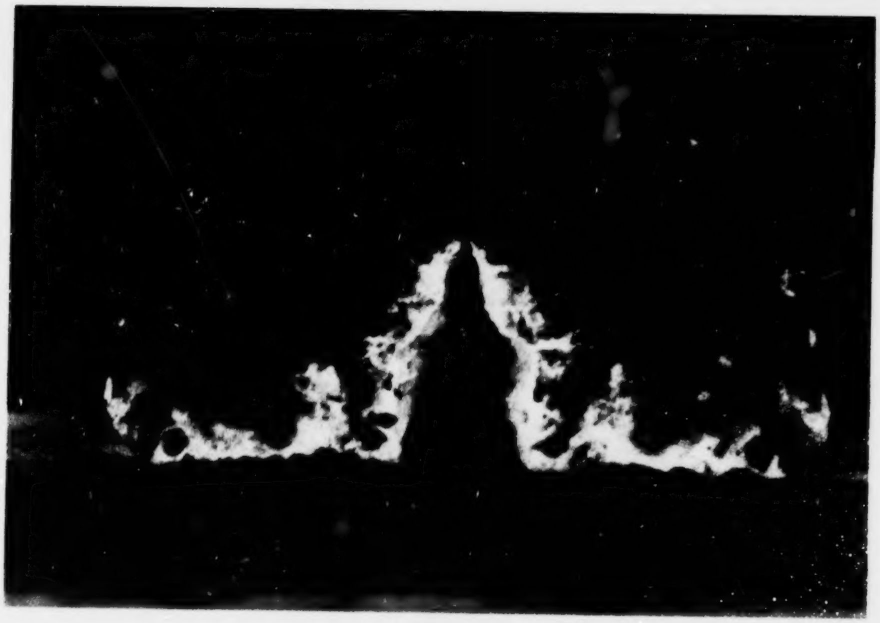


Photograph 49
TR24 Sample 1 - Exposure 10 days

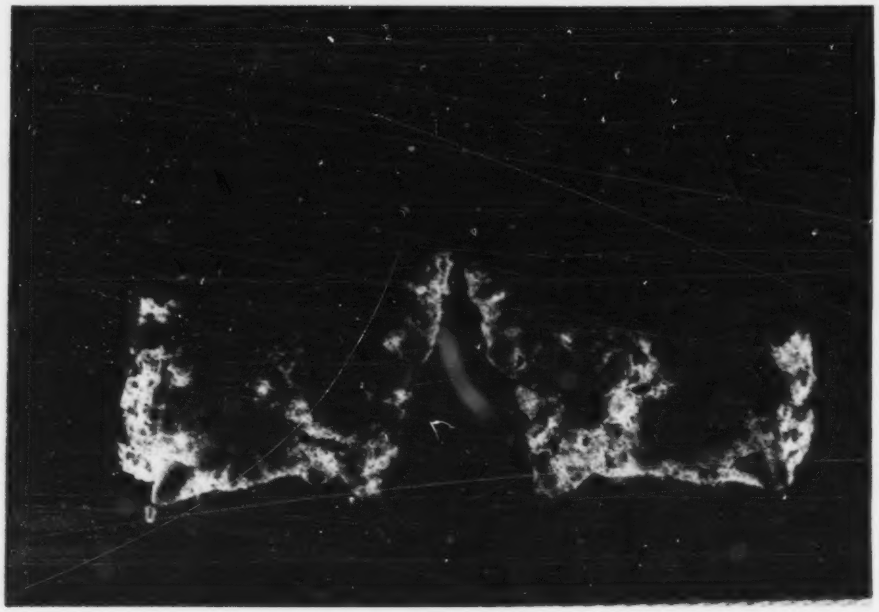
Photograph 50
TR24 Sample 1 - Exposure 21 days



Photograph 51
TR24 Sample 2 - Exposure 1 day



Photograph 52
TR24 Sample 2 - Exposure 4 days



Photograph 53
TR24 Sample 2 - Exposure 10 days

Photograph 54
TR24 Sample 2 - Exposure 21 days

Photograph 55
TR24 Sample 3 - Exposure 1 day

Photograph 56
TR24 Sample 4 - Exposure 1 day

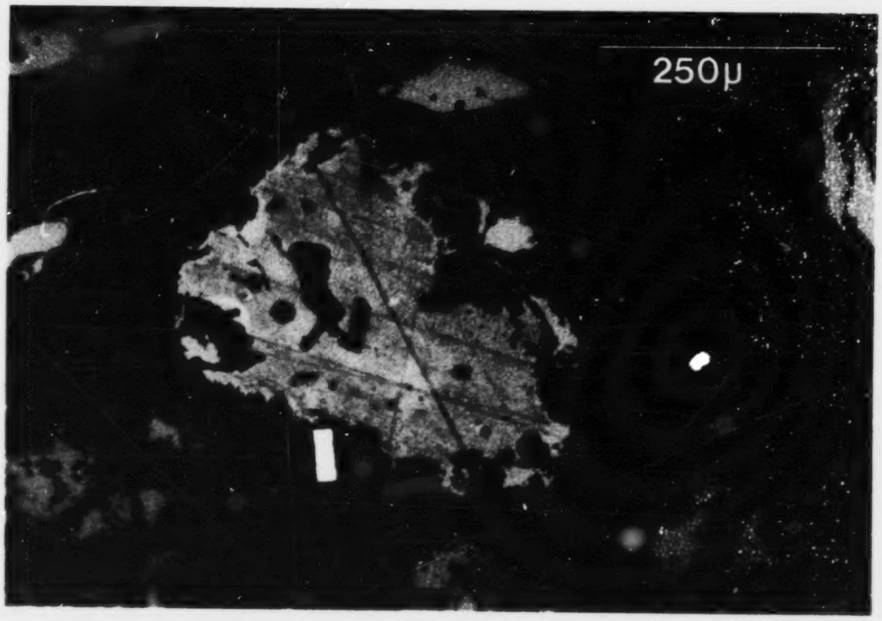
Photograph 57
TR24 Sample 4 - Exposure 4 days

Photograph 58
TR24 Sample 4 - Exposure 10 days

Photograph 59
TR24 Sample 4 - Exposure 21 days

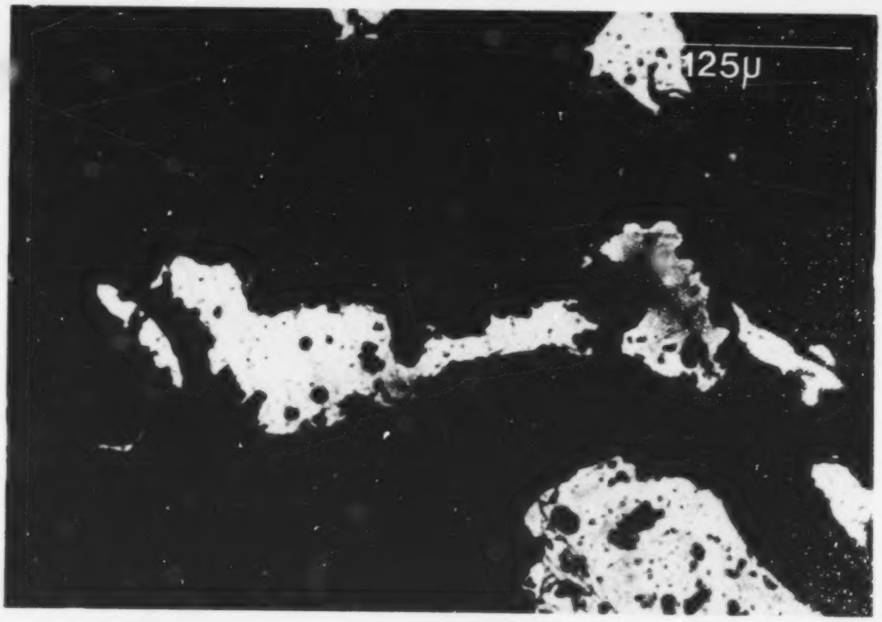
APPENDIX II

Photomicrographs



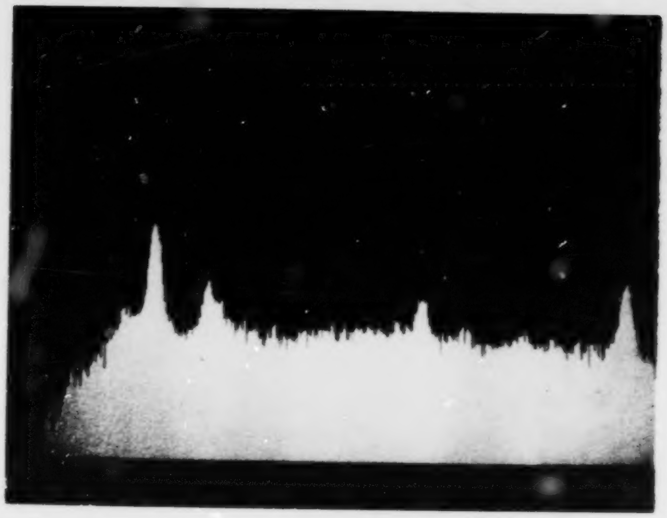
Photograph 60. 100 X (Reflected Light)

Coarse free particle of slightly argentiferous gold with minor internal pitting and peripheral lenticles or encrustations. Gold:silver ratio constant; zoning absent. Gangue particles are primarily magnetite-ilmenite.

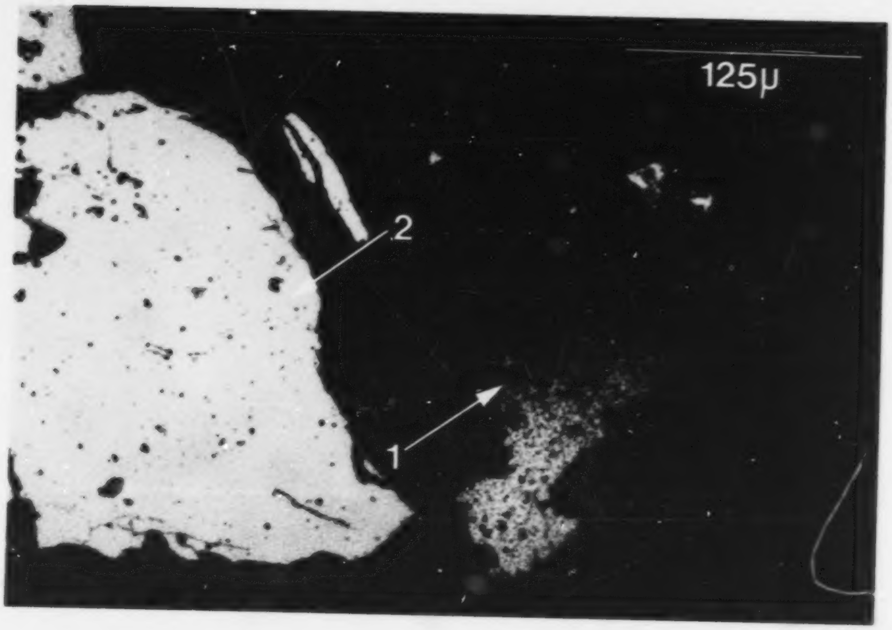


Photograph 61. 200 X (Reflected Light)

Irregular elongate, slightly argentiferous free gold particle. Reflections indicate considerable depth extent.

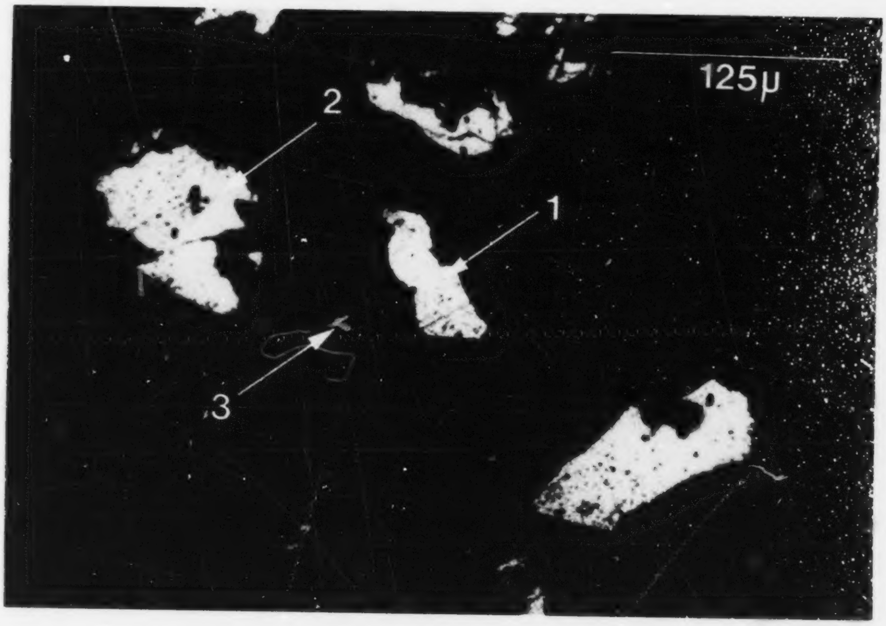


Photograph 62. SEM-EDXA spectra.
Log scale. Slightly argentiferous gold.



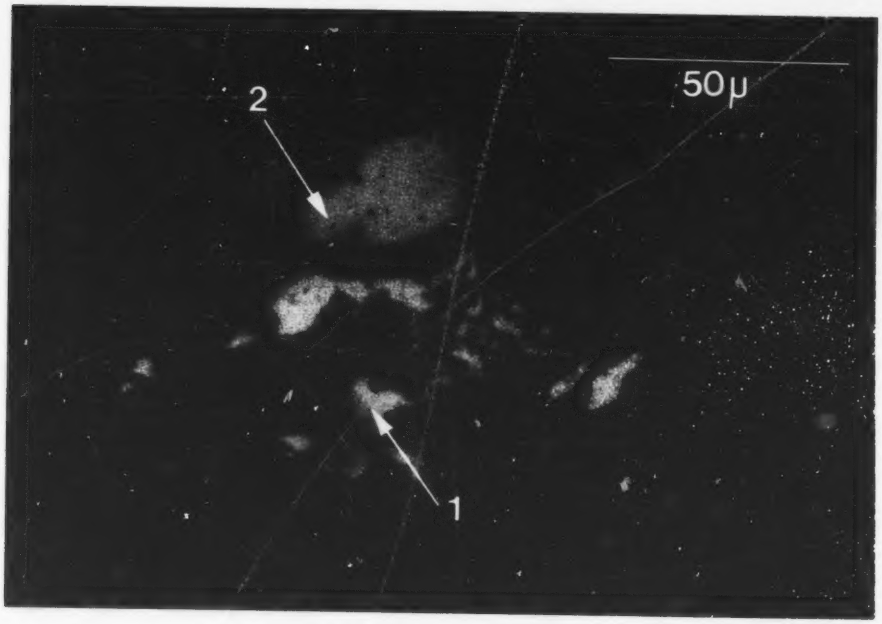
Photograph 63. 200 X (Reflected Light)

Irregular free particle of gold (1) next to coarse anhedral arsenopyrite (2). Tarnish due to surface oxidation of polished mount.



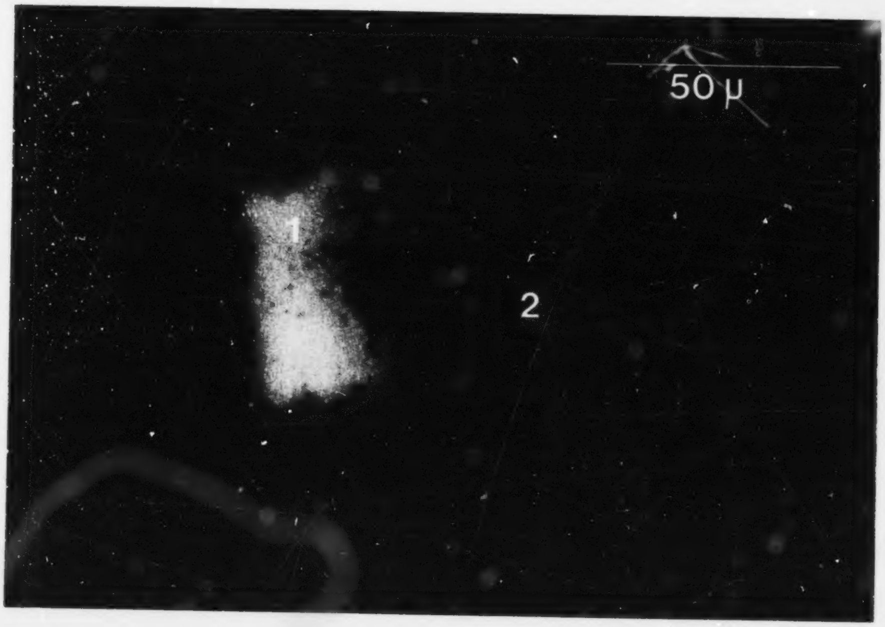
Photograph 64. 200 X (Reflected Light)

Smooth free particle of slightly argentiferous gold (1) with arsenopyrite (2) and magnetite-ilmenite (3).



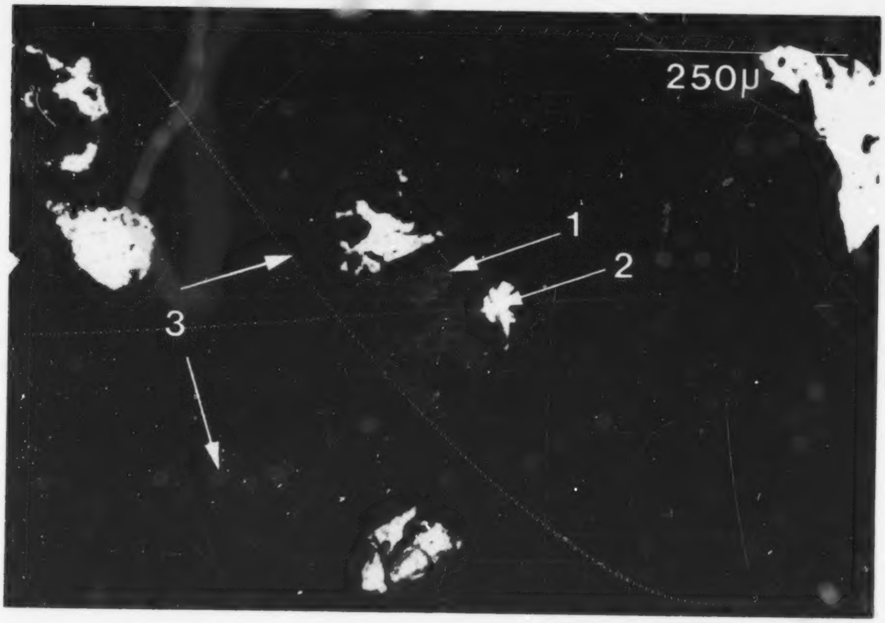
Photograph 65. 500 X (Reflected Light)

Irregular lenticle-type slightly argentiferous free gold particle (1) adjacent to arsenopyrite (2).



Photograph 66. 500 X (Reflected Light)

Angular middling particle of weakly argentiferous gold (1) adjacent to iron hydroxide (2).



Photograph 67. 100 X (Reflected Light)

Middling association between ilmenite (1) and small particle of slightly argentiferous gold (2). Note layering in iron oxyhydroxides (3)

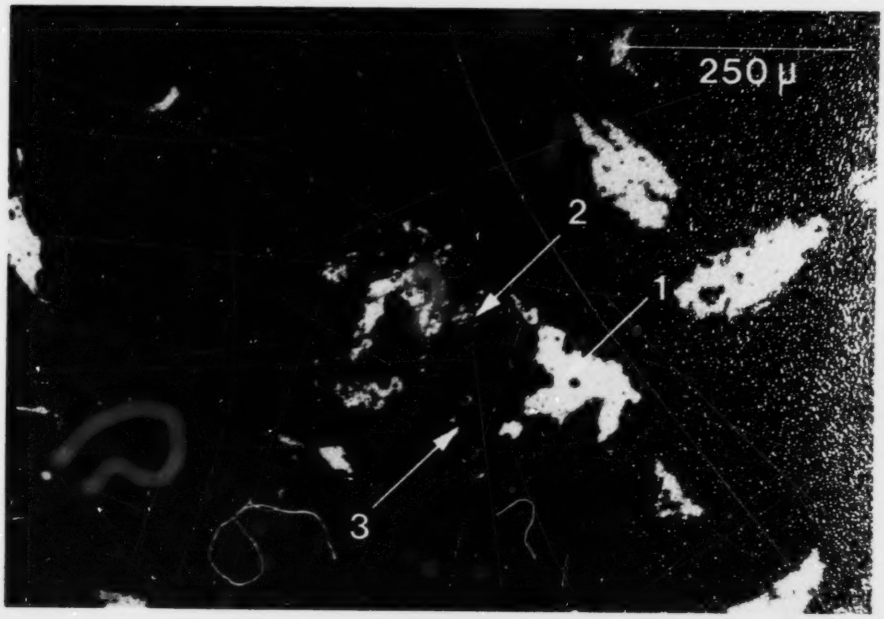


Photo 68. 100X (Reflected Light)

Middling association of irregular, slightly argentiferous gold (1)and irregular, crustiform-type particle of native copper (2).

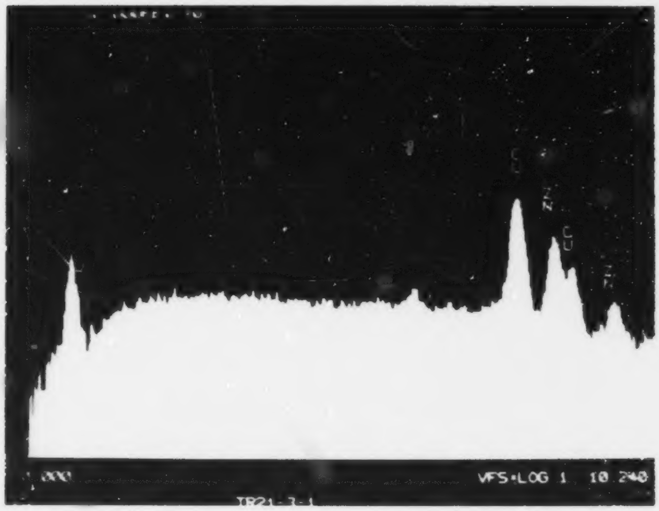


Photo 69. SEM-EDXA Spectra.
Log scale. Particle of native copper.

Photo 70. 1250X (Reflected Light-Oil Immersion).

Fine particle of argentiferous gold (1)
enclosed by pyrite (2).

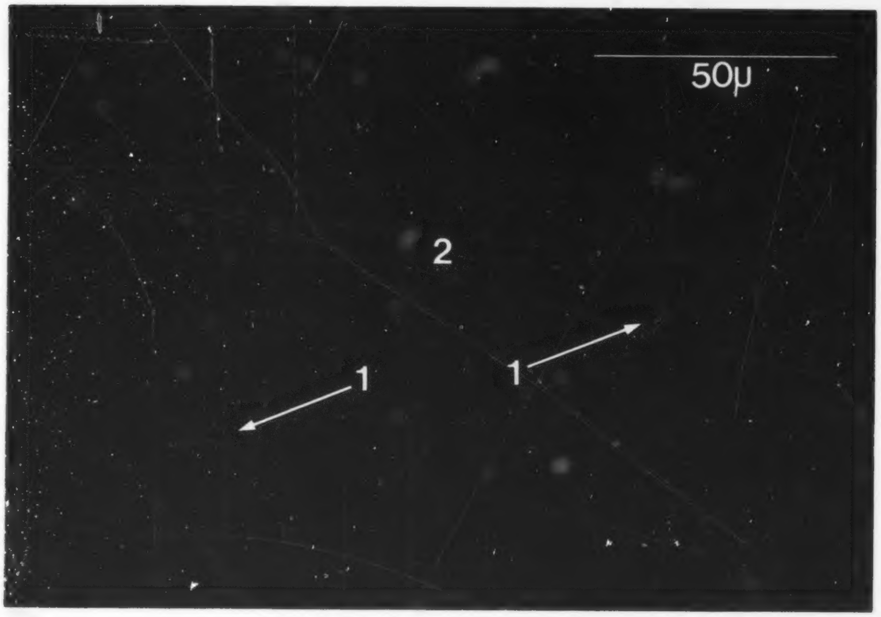


Photo 71. 500X (Reflected Light).

Two minute particles of argentiferous gold (1)
locked within pyrite (2).

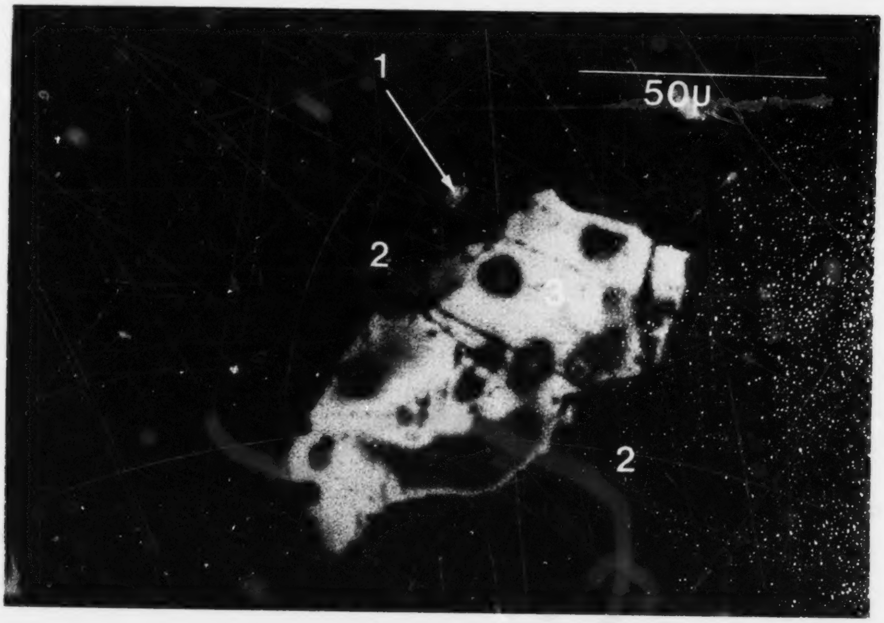


Photo 72. 500X (Reflected Light)

Minute particle of native gold (1) enclosed by secondary iron hydroxides (2) which replace pyrite (3).

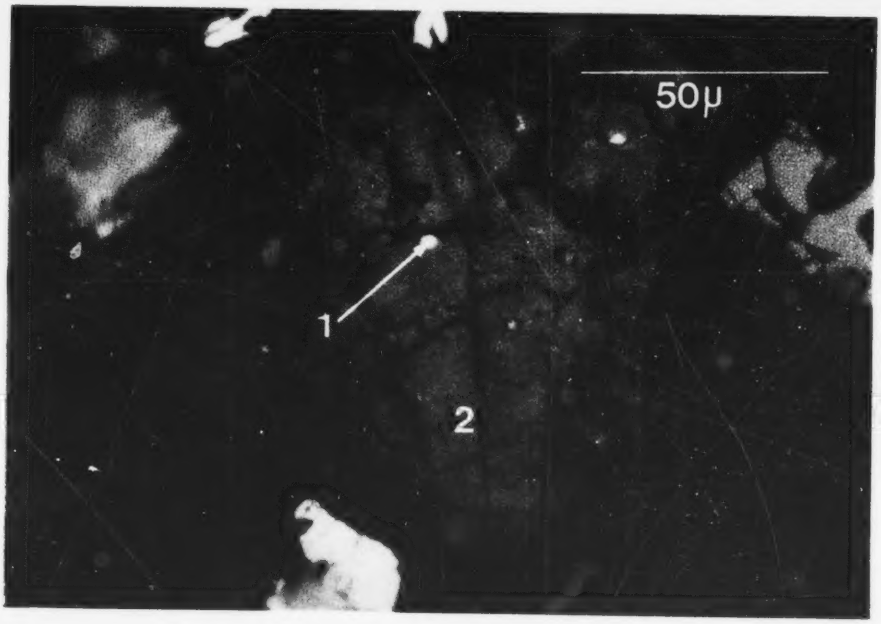


Photo 73. 500X (Reflected Light)

Minute particles of native gold (1) adjacent to fracture within secondary iron hydroxide (2) after iron sulphide.

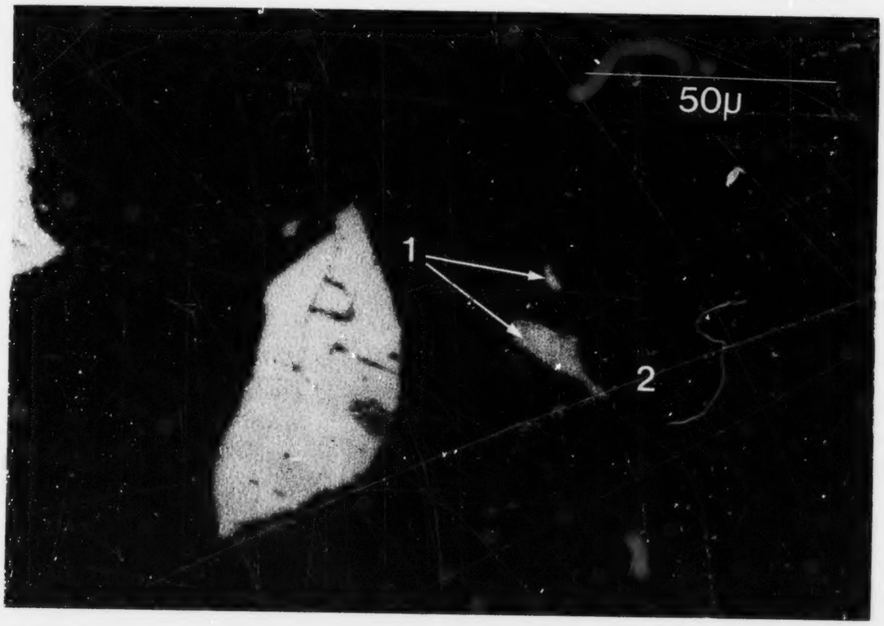


Photo 74. 500X (Reflected Light)

Several minute particles of gold (1) enclosed by porous secondary iron hydroxide (2).

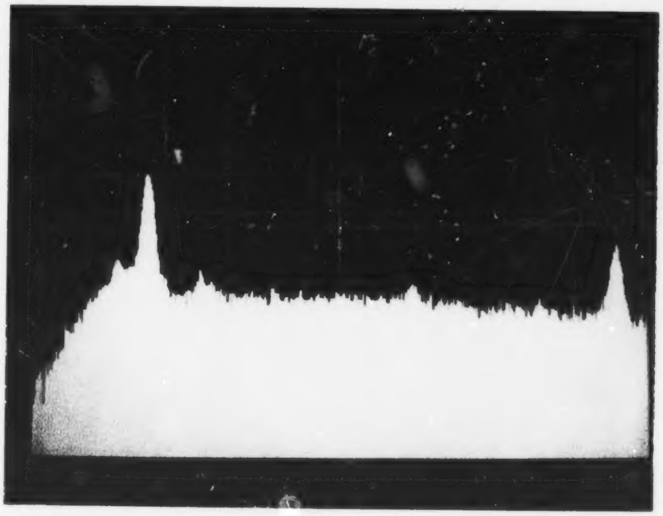


Photo 75. SEM-EDXA spectra.

Log scale. Particle of high fineness gold.

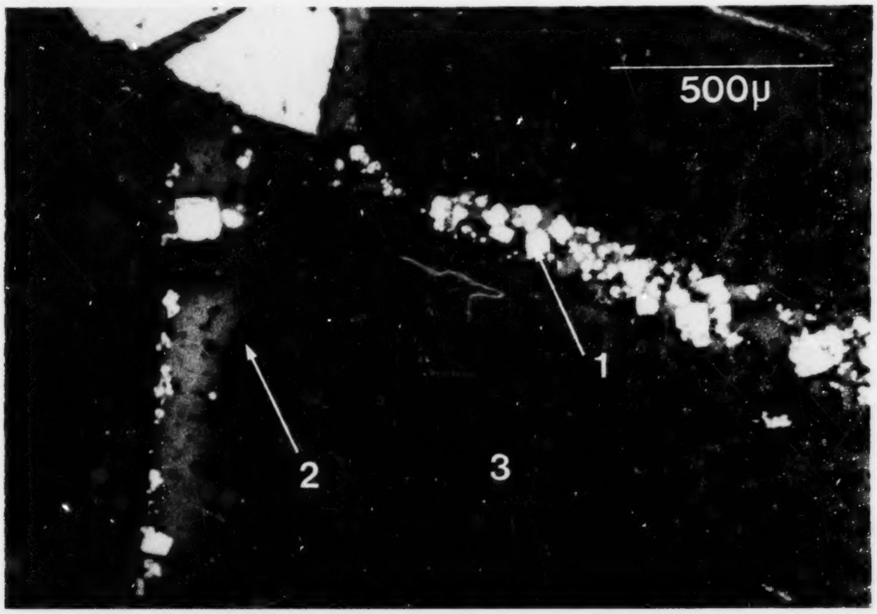


Photo 76. 50X (Reflected Light)

Fracture controlled sulphide mineralization, chiefly pyrite (1) within quartz-white mica-illite veinlets (2) enclosed by quartz of feldspathic groundmass (3).

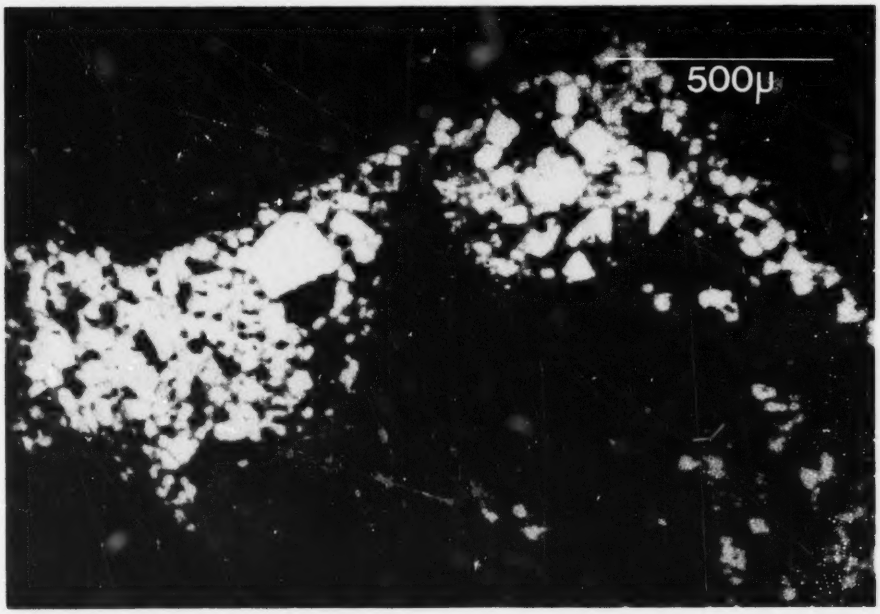


Photo 77. 50X (Reflected Light)

Dilatant zones at fracture intersections exhibit abundant sulphide mineralization, chiefly pyrite, arsenopyrite and lesser oxidized replacements of the sulfides.

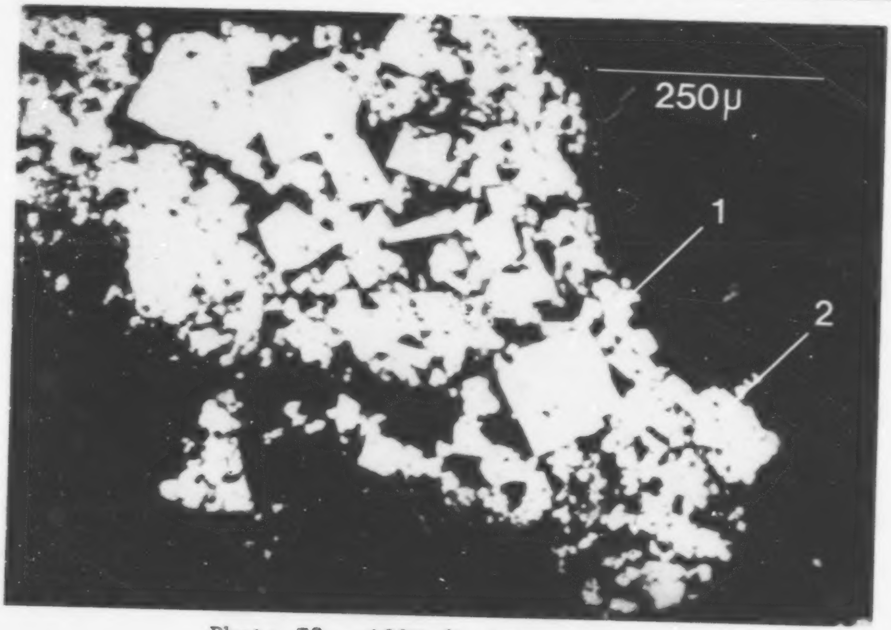


Photo 78. 100X (Reflected Light)

Coarse aggregate of pyrite crystals (1) euhedral to subhedral in form, in lenticular dilation along sericitized (2) fracture zone enclosed by quartzofeldspathic groundmass.

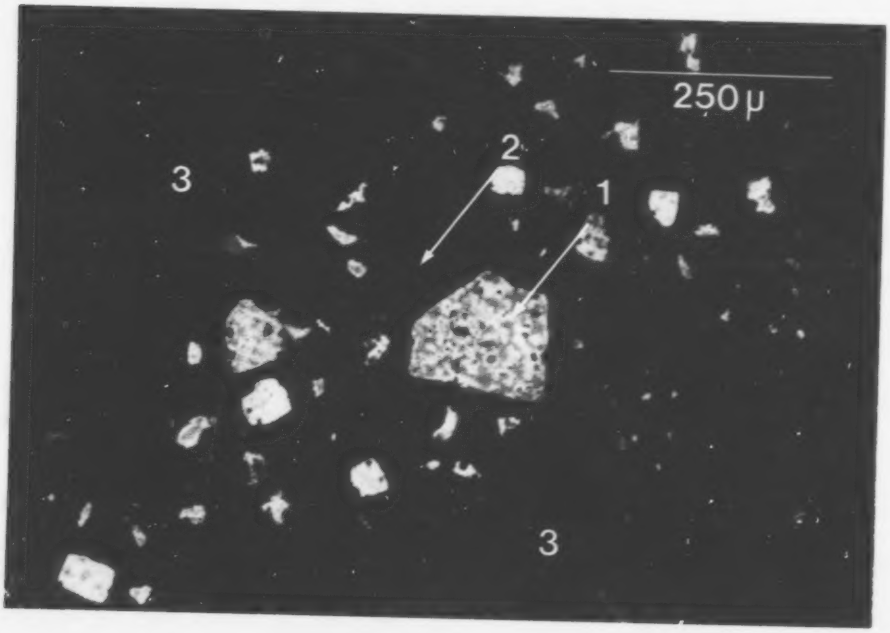


Photo 79. 100X (Reflected Light)
Aggregate of euhedral to subhedral pyrite (1) exhibit incipient to pervasive oxidation to secondary oxides and hydroxides (2). Note laminated fracture and peripheral control of oxide replacement. Sulfides enclosed by quartz (3) with interstitial illite-sericite (4).

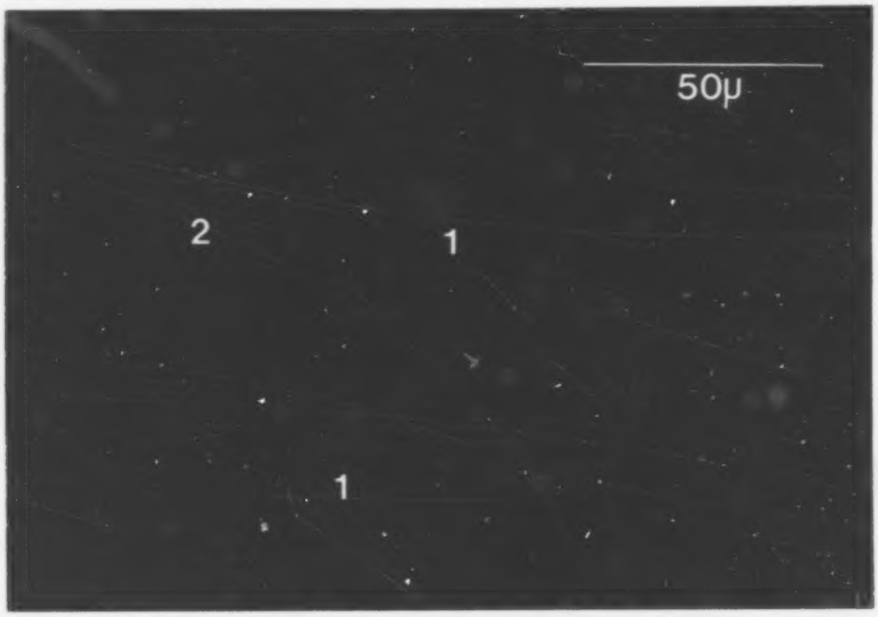


Photo 80. 500X (Reflected Light)

Pyrite (1) exhibits peripheral oxidation replacement by encrustation laminated secondary iron oxyhydroxides (2).

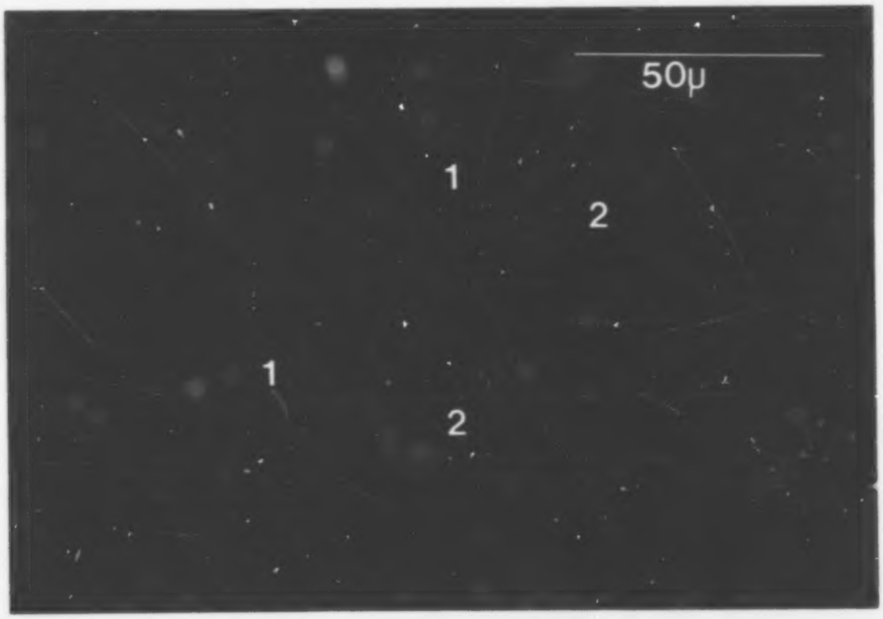


Photo 81. 500X (Reflected light)

Strong fracture controlled replacement of pyrite (1) by layered secondary iron oxyhydroxides (2).

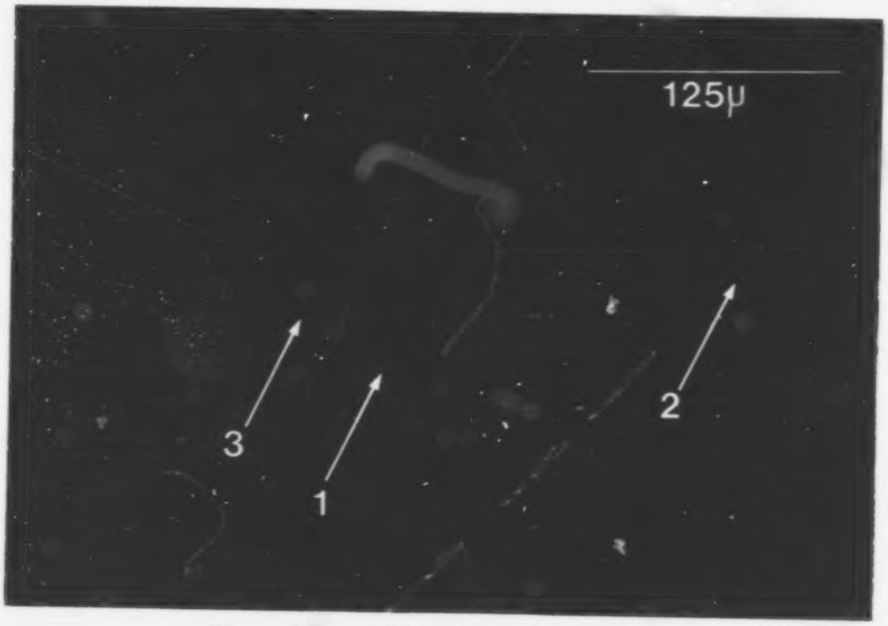


Photo 82. 200X (Reflected Light)

Oxidation to secondary iron oxyhydroxides of pyrite (1). Hydroxides form anhedral laminated collomorphic (2) or pseudomorphic poorly laminated types (3).

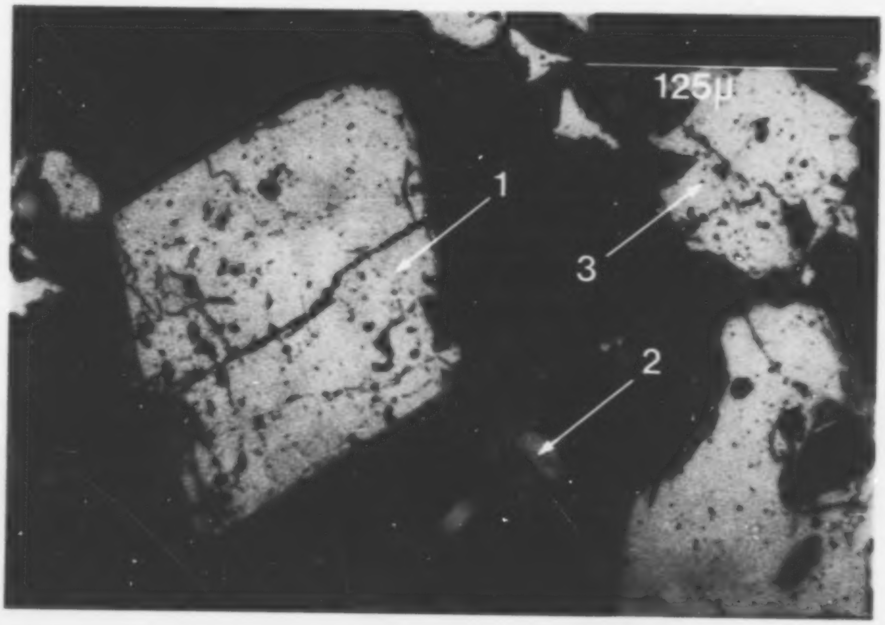


Photo 83. 200X (Reflected light)

Anhedral crystal of arsenopyrite (1), subhedral chalcopyrite (2) and partially oxidized pyrite (3).

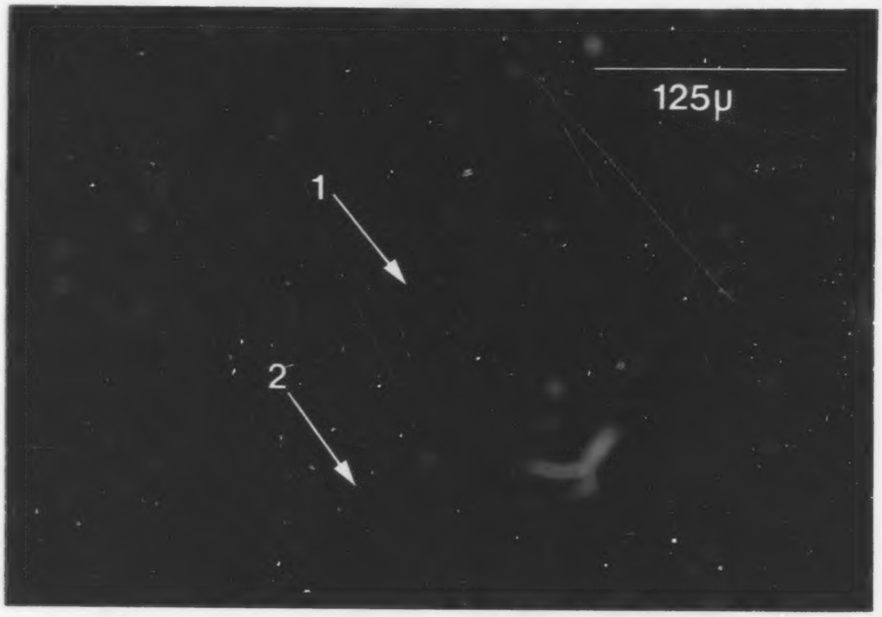


Photo 84. 200 X (Reflected Light)

Aggregate of euhedral to subhedral pyrite crystals (1) cemented by chalcopyrite (2).

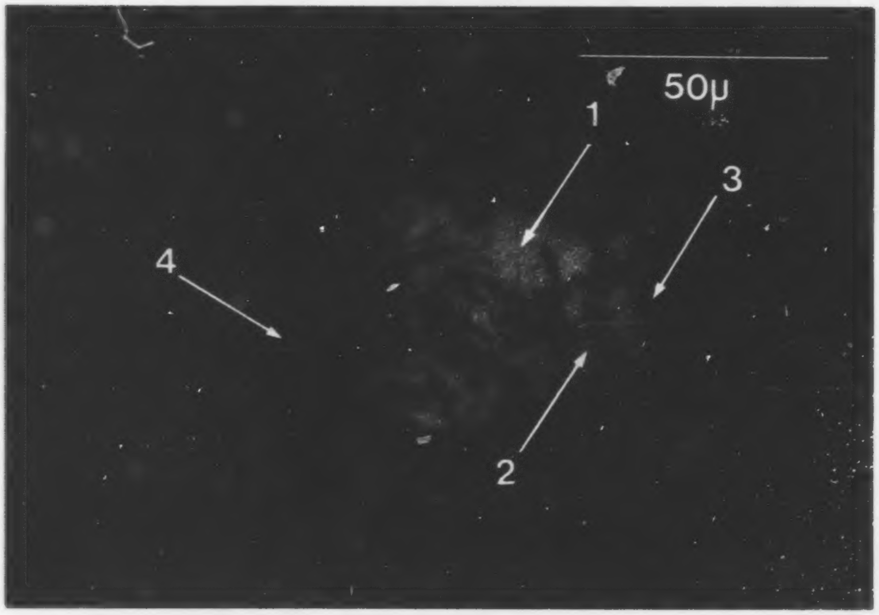


Photo 85. 500X (Reflected Light)

Free particles of chalcopyrite (1) with inclusions of galena (2) and peripheral fracture controlled oxidation to chalcocite (3) and covellite (4).

WD1

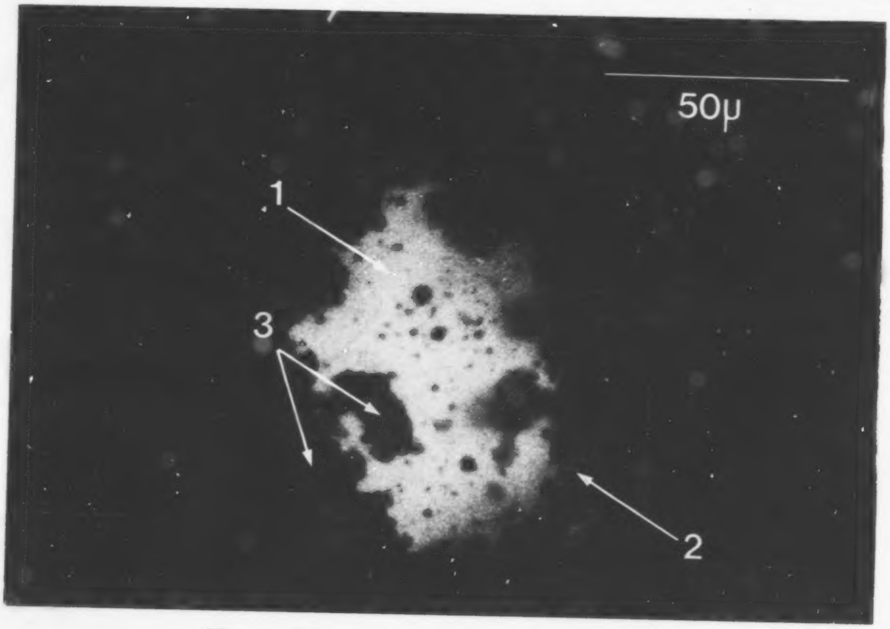


Photo 86. 500X (Reflected Light)

Free particle of chalcopyrite (1) with peripheral native copper (2), minor sphalerite and secondary iron hydroxides (3).

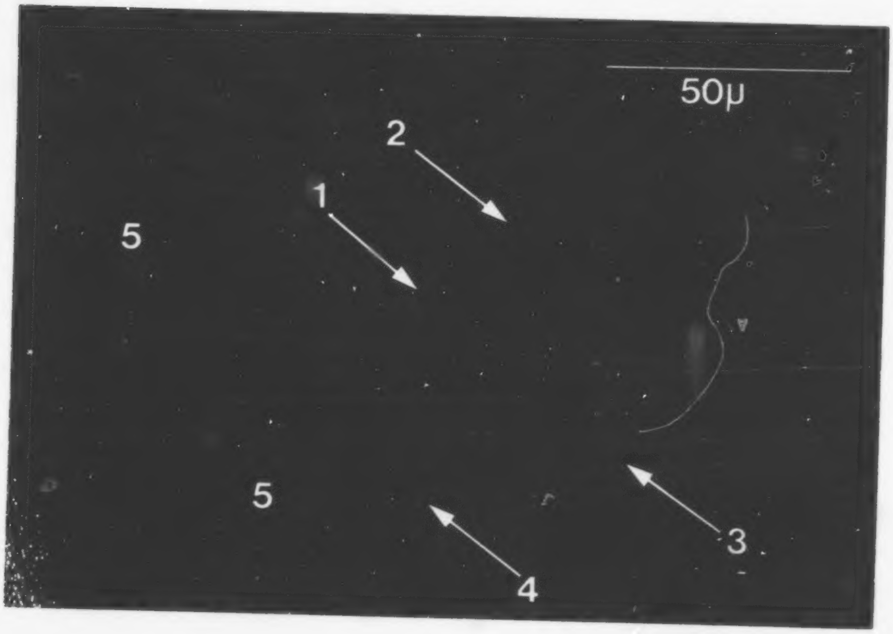


Photo 87. 500X (Reflected Light)

Irregular inclusions of chalcopyrite (1), pyrrhotite (2), rutile (3) and sphalerite with exsolved chalcopyrite (4) within aggregate of pyrite (5). Base metals may cement pyrite subhedra.

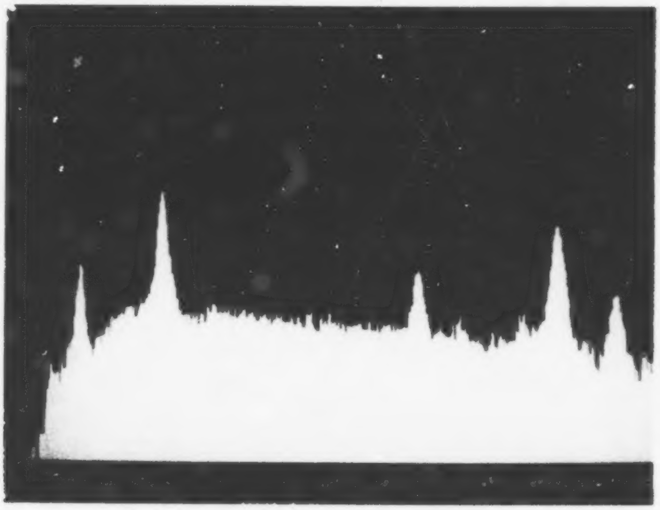


Photo 88. SEM-EDXA Spectra.
Log scale. Sphalerite - ZnS.

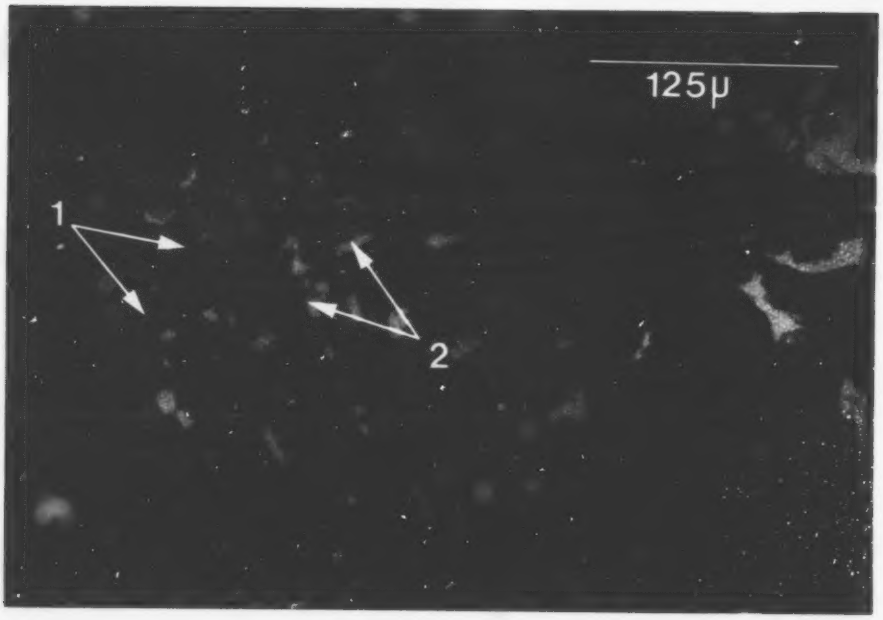


Photo 89. 200X (Reflected Light)

Coarse free particle of sphalerite (1) with crystallographically-controlled bleb-type exsolution of chalcopyrite (2).

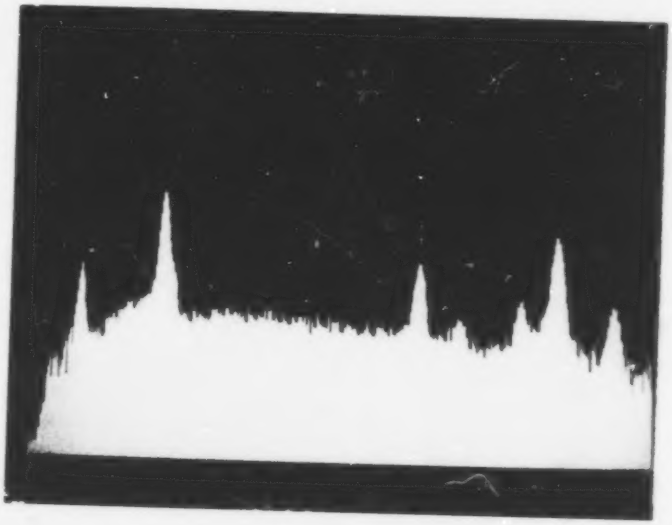


Photo 90. SEM-EDXA Spectra.
Log scale. Sphalerite (ZnS) with exsolved chalcopyrite ($CuFeS_2$).

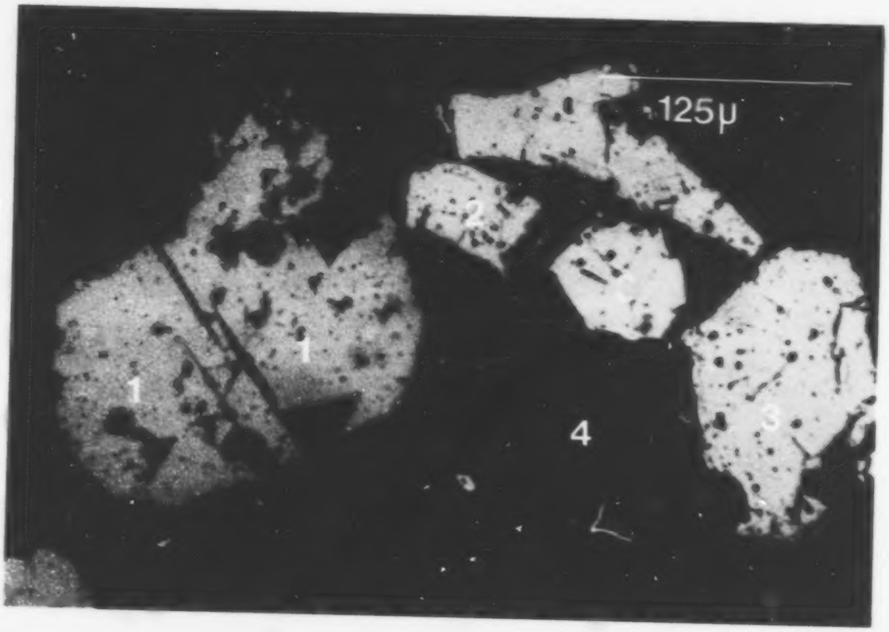


Photo 91. 200X (Reflected Light)

Coarse free particle of galena (1) exhibits typical triangular pits. Pyrite (2), arsenopyrite (3) and magnetite (4) are also observed.

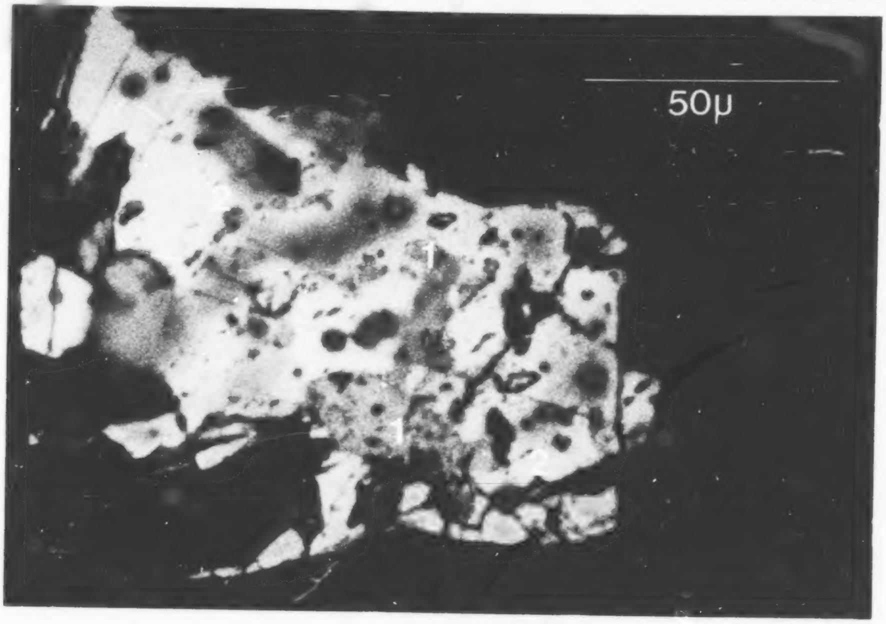


Photo 92. 500X (Reflected Light)

Numerous irregular inclusions of galena (1) with fractured pyrite (2).

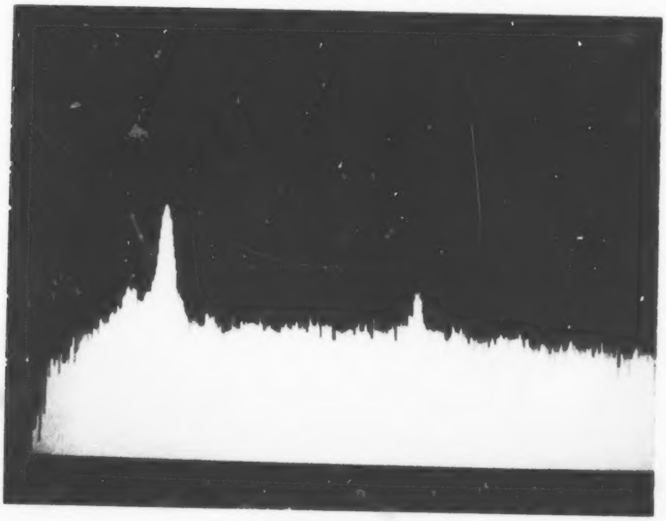


Photo 93. SEM-EDXA Spectra

Log scale. Galena- PbS.

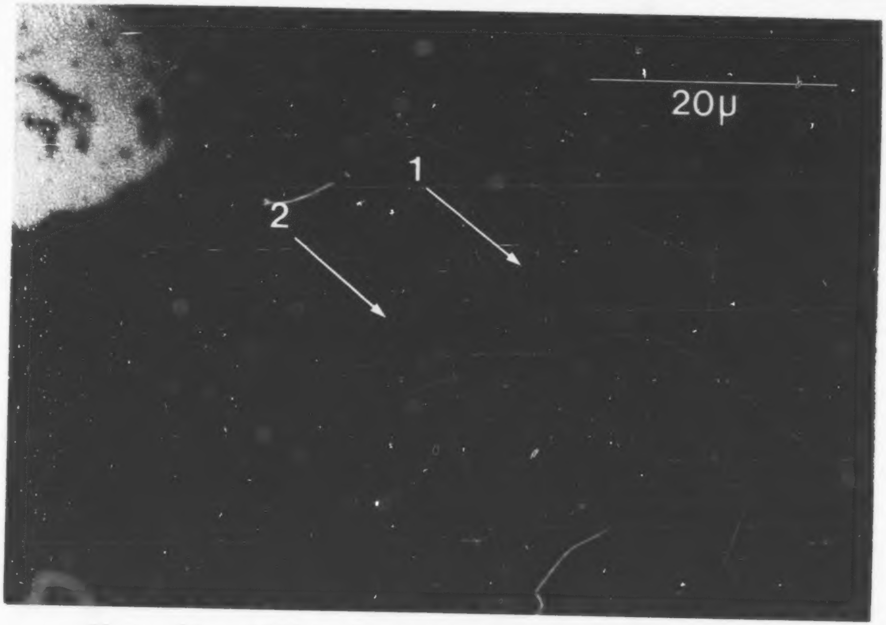


Photo 94. 1250X (Reflected Light-Oil Immersion)

Minute free particle of molybdenite (1) exhibits platy fabric. Secondary iron hydroxides (2) and sphene replace ilmenite-hematite particle.

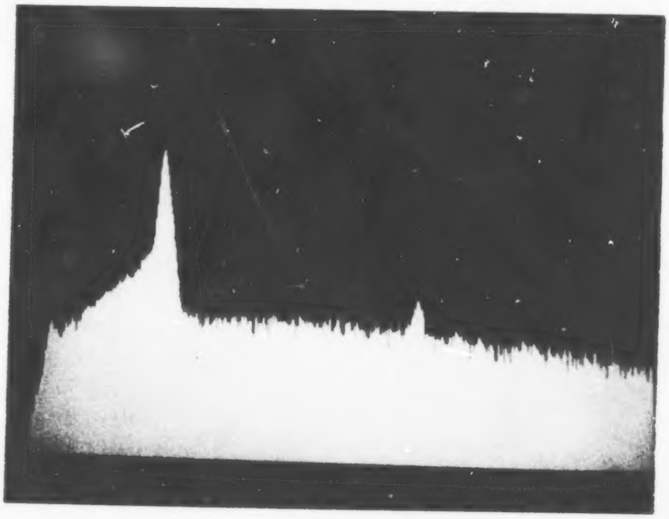


Photo 95. SEM-EDXA Spectra.

Log scale. Molybdenite- MoS_2.

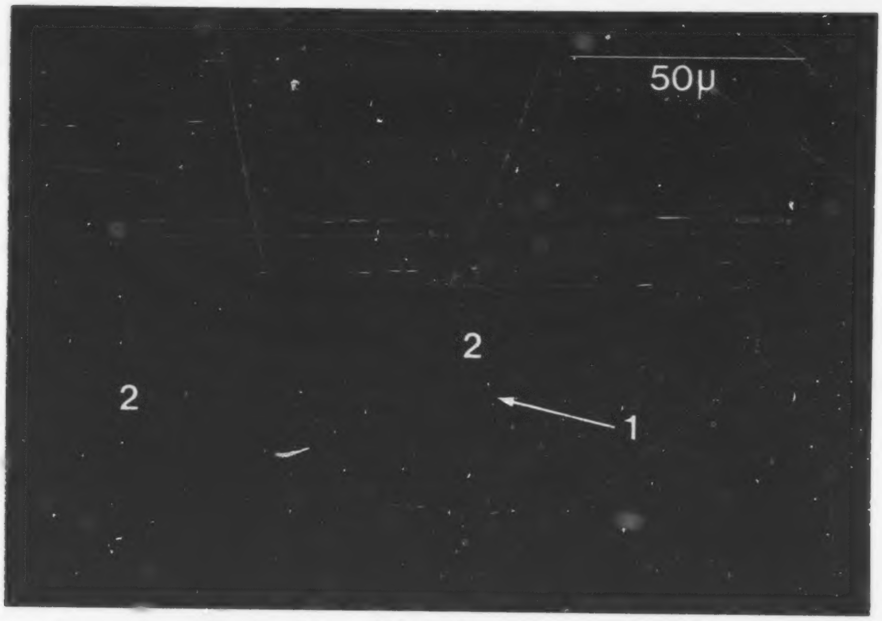


Photo 96. 500X (Reflected Light)

Fine inclusions, tentatively identified as tellurobismuthite (1), within arsenopyrite (2).

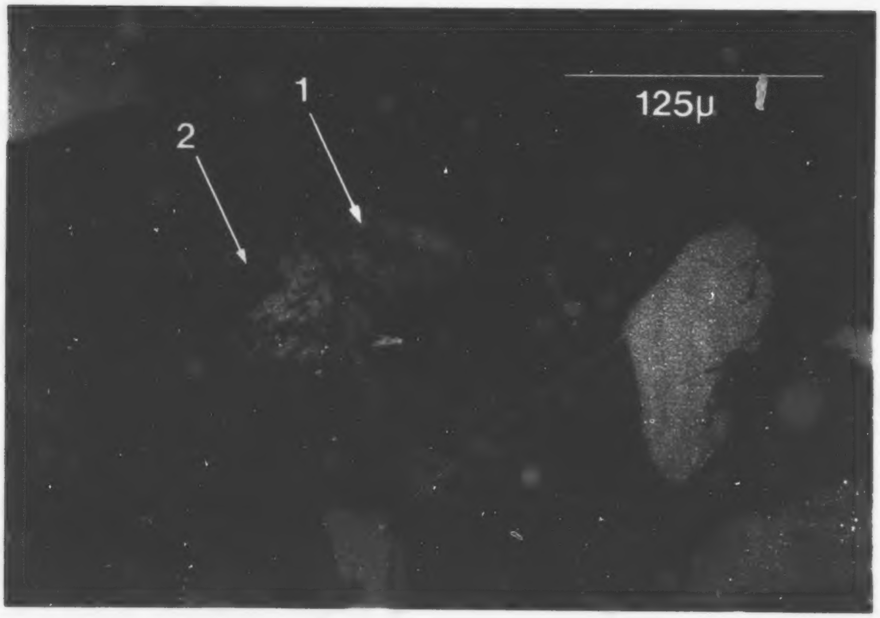


Photo 97. 200X (Reflected Light)

Coarse particle of tabular to sheaf-like pyrolusite (1) coated by iron hydroxides (2).

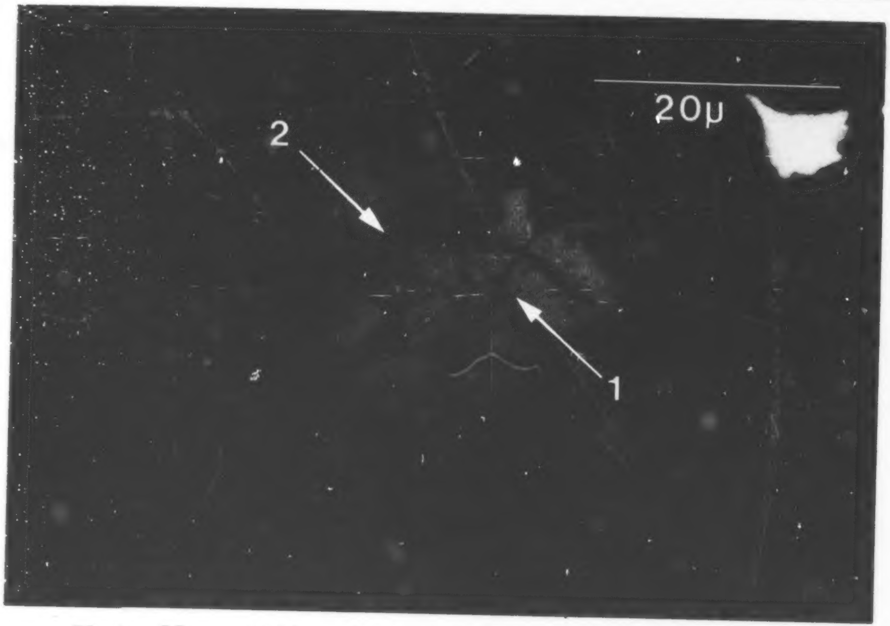


Photo 98. 1250X (Reflected Light - Oil Immersion)

Free particle of manganese oxide, tentatively pyrolusite (1) with marginal iron hydroxides (2).

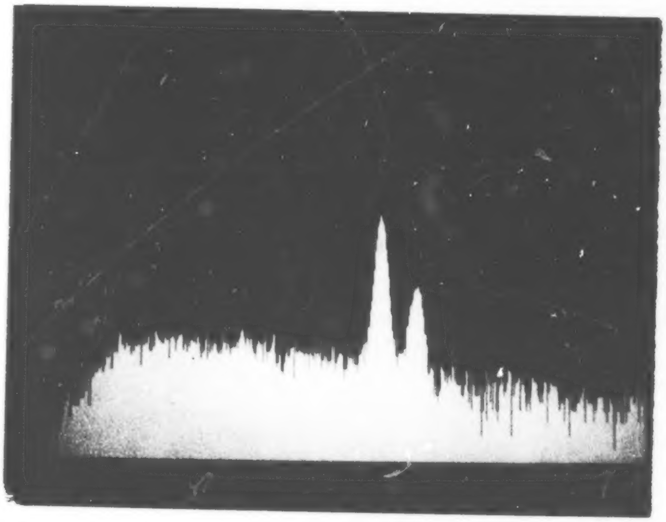


Photo 99. SEM-EDXA Spectra.

Log Scale. Pyrolusite MnO_2

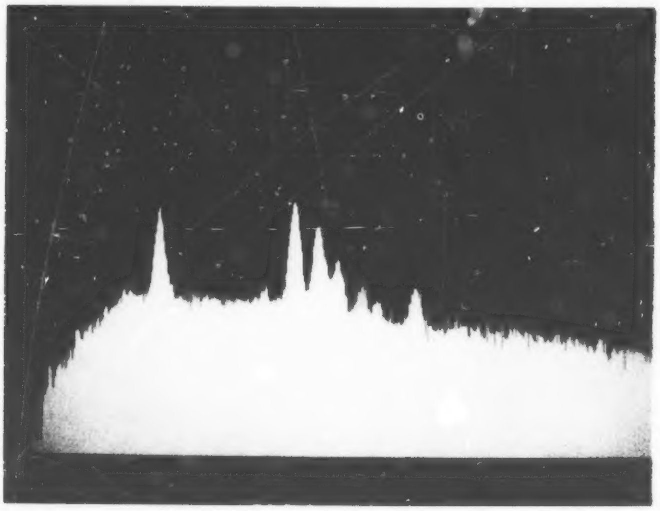


Photo 100 - SEM-EDXA Spectra.

Log Scale. Barite - $BaSO_4$

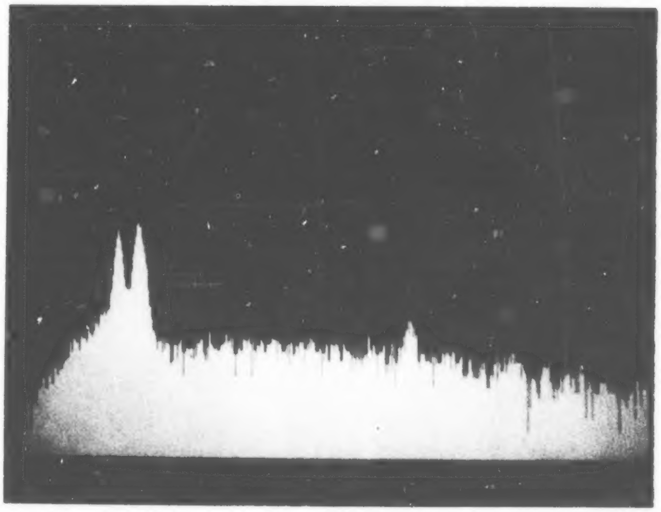


Photo 101. SEM-EDXA Spectra.

Log Scale. Zircon- $ZrSiO_4$

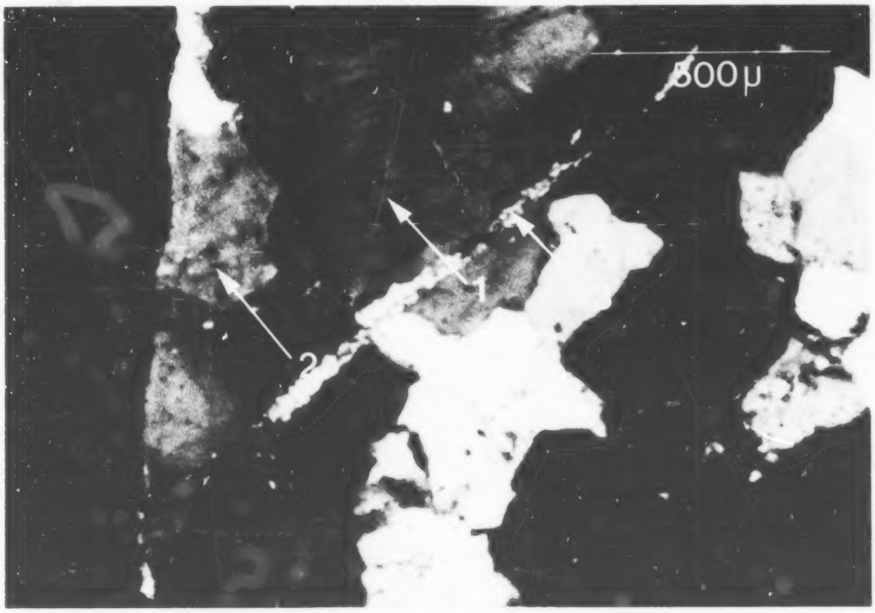


Photo 102. 50X (Transmitted Light - Crossed Nicols)

Aggregate of quartz (1), lesser potash feldspar (orthoclase/adularia) (2) cut by veinlet of muscovite-sericite-illite (3).

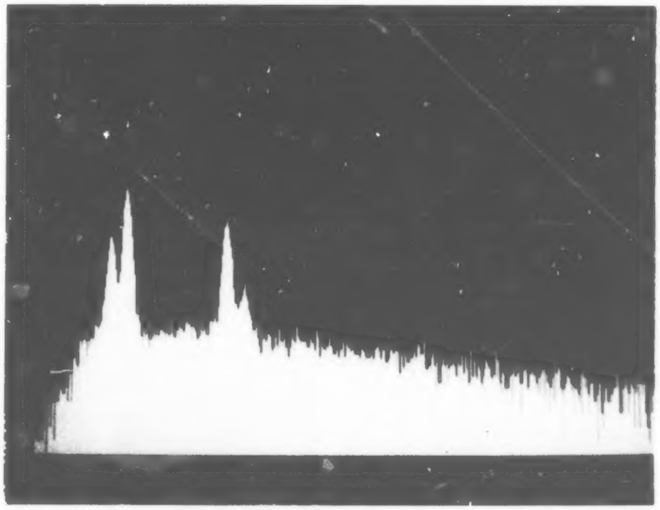


Photo 103. SEM-EDXA Spectra.

Log Scale. Potash-feldspar (orthoclase-adularia) $KAlSi_3O_8$

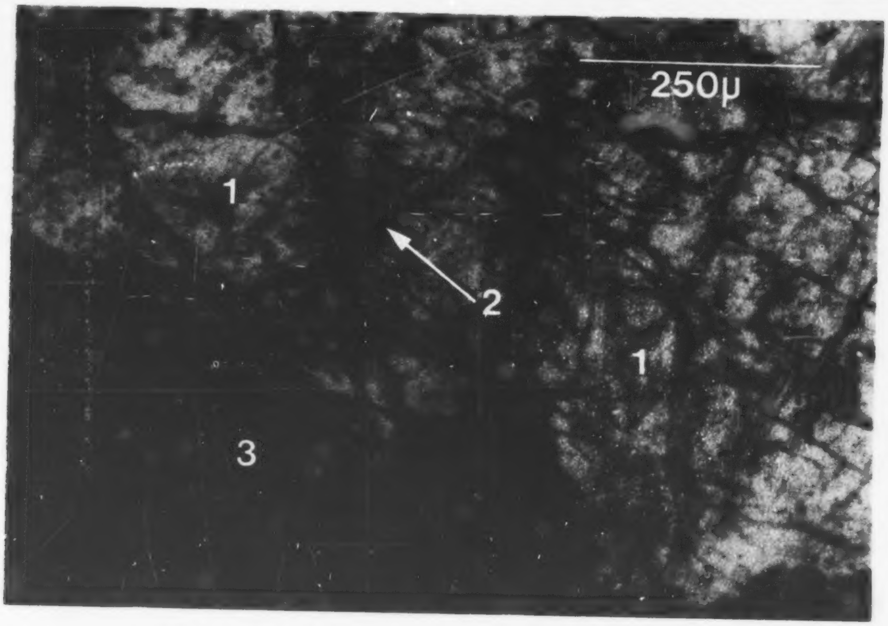


Photo 104. 100X (Transmitted Light)

Aggregate of quartz (1) fractured and veined by Fe-stained sericite-illite (2) adjacent to kaolinized feldspar (3).

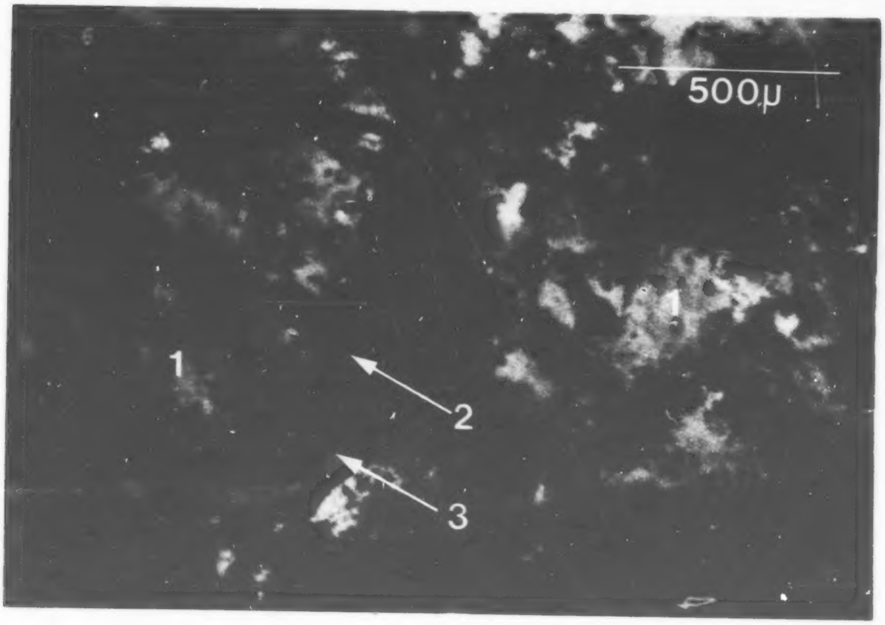


Photo 105. 50X (Transmitted Light - Crossed Nicols)

Quartzose aggregate (1) exhibits fracture filling by chlorite (anomalous blue coloration) (2) and lesser illite-sericite (3).

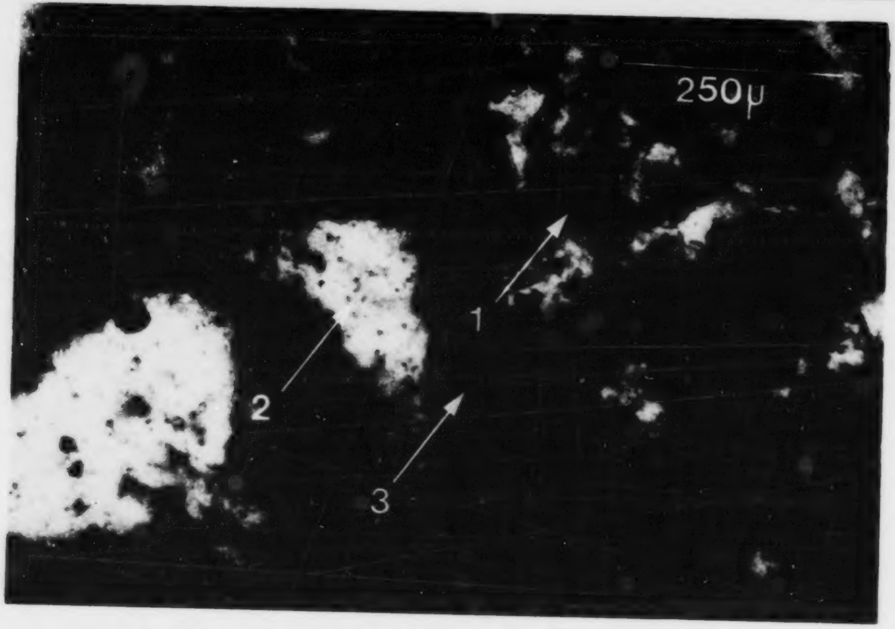


Photo 106. 100X (Transmitted Light)

Alteration and replacement of primary quartzofeldspathic groundmass by Fe-Mg chlorite (1) quartz (2) and cloudy illite-muscovite intergrown with chlorite (3).

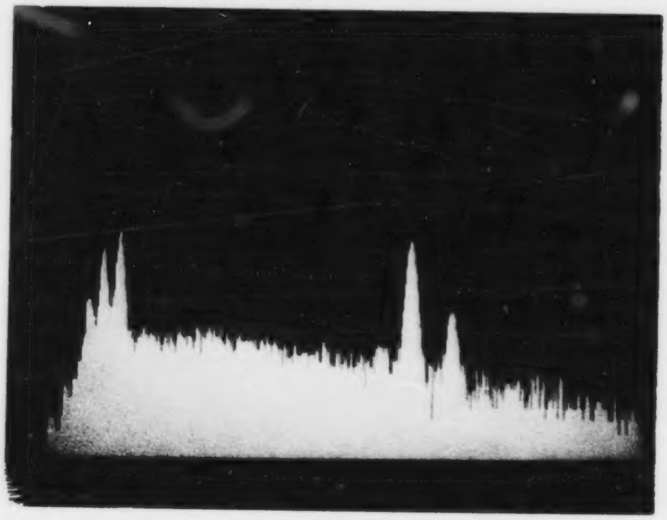


Photo 107. SEM-EDXA Spectra.

Log scale. Fe-Mg chlorite. (Fe, Mg, Al)$_6$ (Si,Al)$_4$O$_{10}$ (OH)$_8$

Photo 108. 100X (Transmitted Light)

Aggregate of chlorite (1), quartz (2) and illite-sericite stained by very fine iron oxyhydroxides (3).

Photo 109. 200X (Transmitted Light)

Pyrite crystals, pseudomorphed by iron hydroxides (1) within matrix of iron stained illite-sericite (2).

Photo 110. 50X (Transmitted Light)

Particle of iron stained phyllic alteration zone comprising illite-sericite (1) and lesser chlorite cut by later muscovite-filled fractures (2).

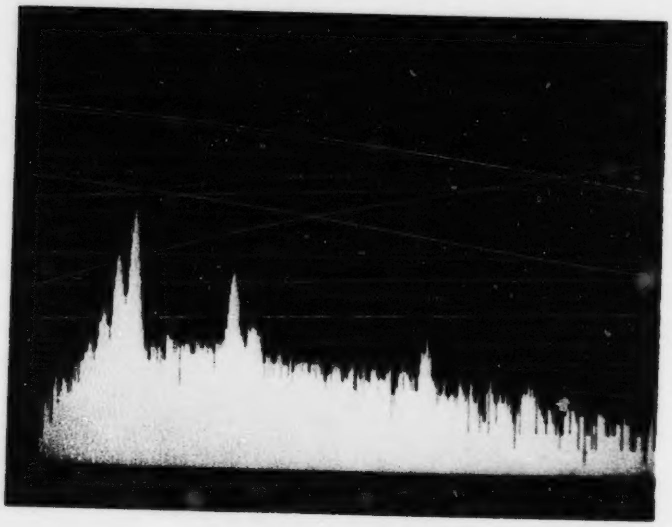


Photo 111. SEM-EDXA Spectra.

Log scale. Chlorite-sericite alteration $(FE,Mg,Al)_6(Si,Al)_4O_{10}(OH)_8-KAl_3Si_3O_{10}(OH)_2$

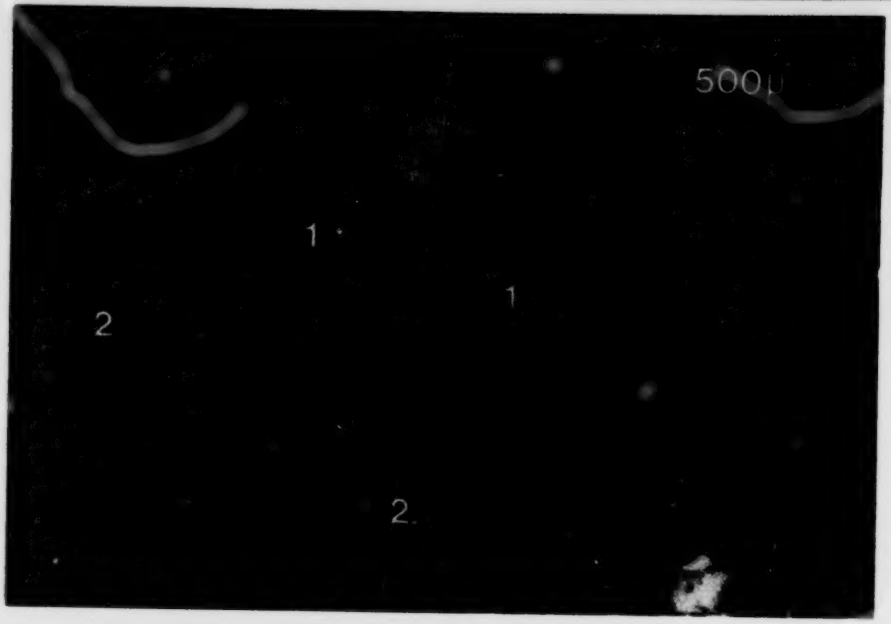


Photo 112. 50X (Transmitted Light-Crossed Nicols)

Quartzose aggregate (1) cut by several sericite-illite fracture veinlets (2).

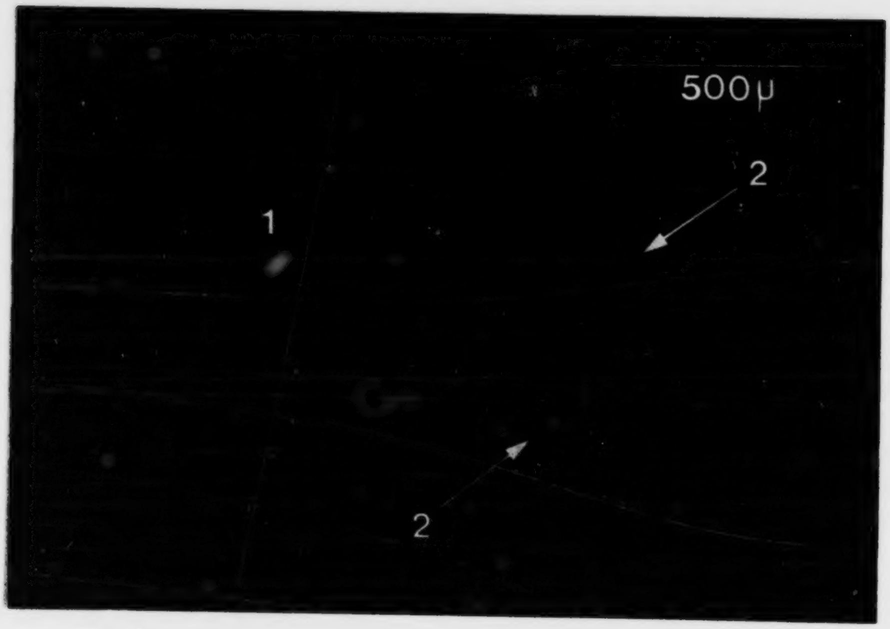


Photo 113. 50X (Transmitted Light)

Relicts of primary volcanic (porphyry) include quartz phenocryst (1) with quartzofeldspathic groundmass partially replaced by clays, chlorite and patchy sericite, cut by sericite-illite filled fractures (2).

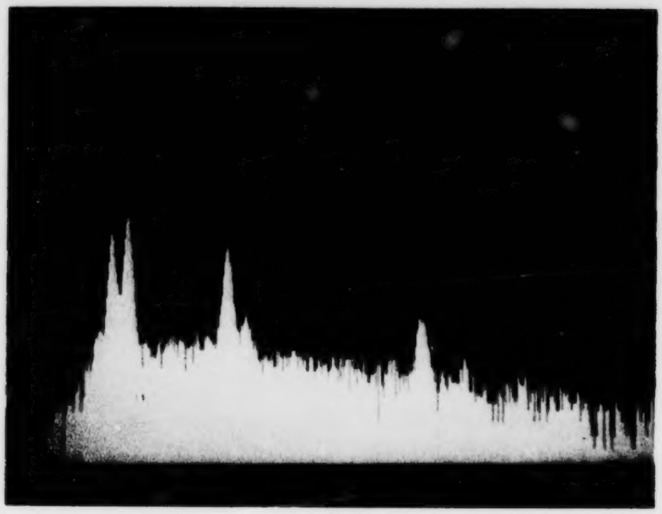


Photo 114. SEM-EDXA Spectra.
Log scale. Sericite-illite stained by Fe Hydroxides.
$KAl_2(OH)_2(AlSi_3O_{10})$ - $KAl_2(OH)_2[AlSi_3(O,OH)_{10}]$

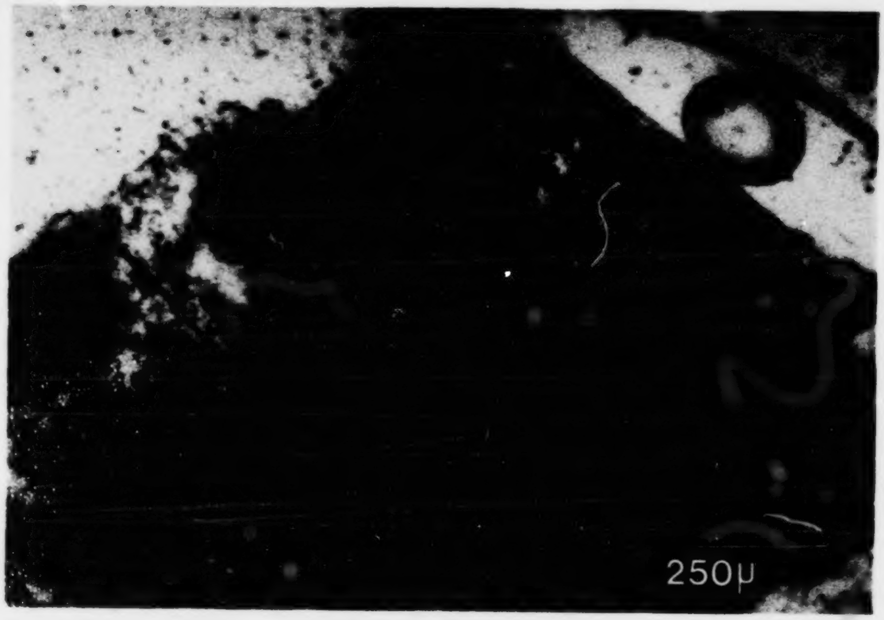


Photo 115. 100X (Transmitted Light)

Early feldspar phenocryst replaced by clay minerals, illite-sericite and Fe hydroxides.

Photo 116. 100X (Transmitted Light - Crossed Nicols)

Graphic intergrowths of quartz and feldspar (1). Feldspars are replaced by Fe-stained illite-sericite. Fractures are filled by clays, mica and fine sulfides.

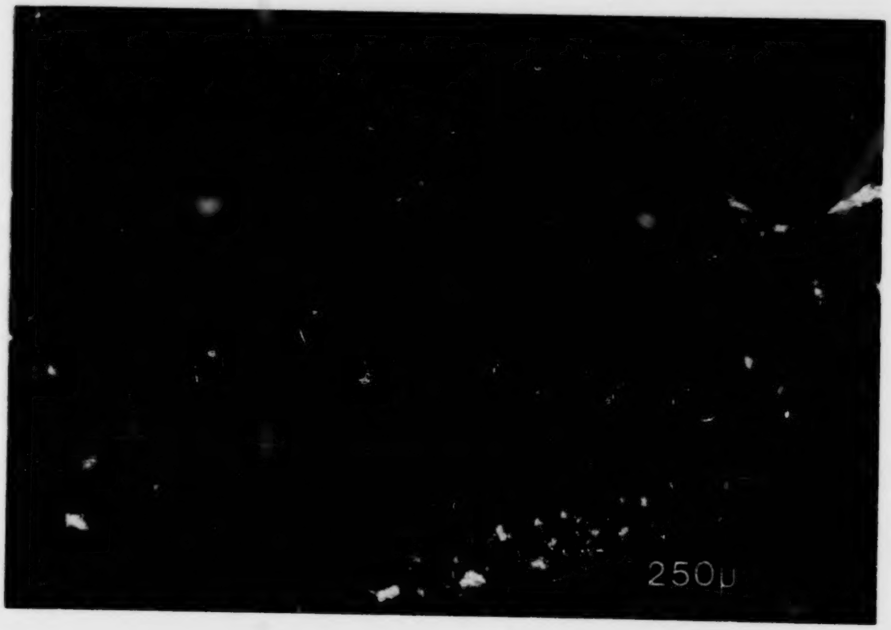


Photo 117. 100X (Transmitted Light - Crossed Nicols)

Feldspar phenocryst replaced by clays, mica and cut by veinlets of Fe hydroxides, sulfides, and sericite.

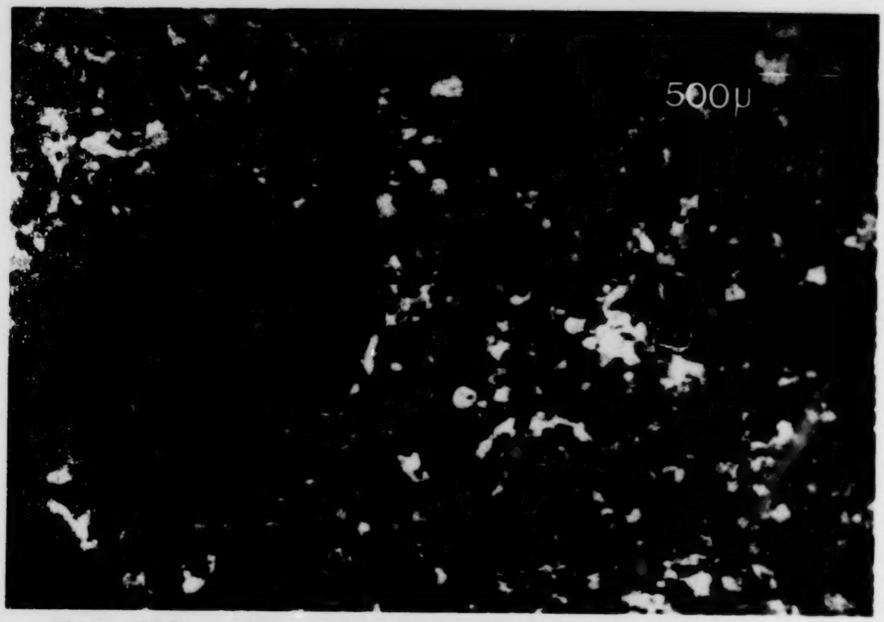


Photo 118. 50X (Transmitted Light- Crossed Nicols)

Relict outline of feldspar phenocryst, replaced by illite-sericite-chlorite, in groundmass of quartz, alkali feldspar and sericite-illite.

Phot 119. 100X (Transmitted Light - Crossed Nicols)

Relict twinning of oligoclase-calcic feldspar, locally replaced by illite-sericite,, iron hydroxides and chlorite.

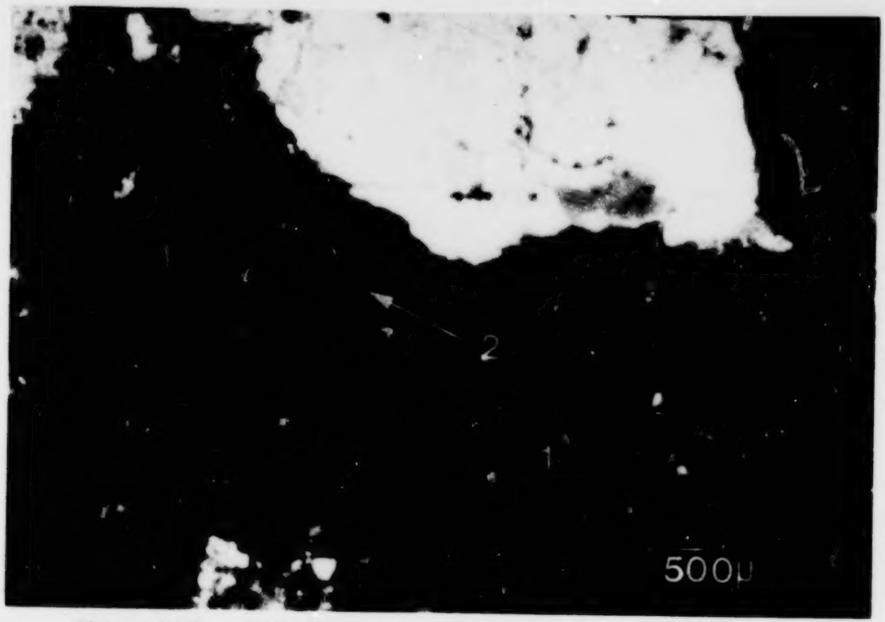


Photo 120. 50X (Transmitted Light- Crossed Nicols)

Quartz grain (1) crosscut by several illite-sericite-Fe hydroxide veinlets (2)

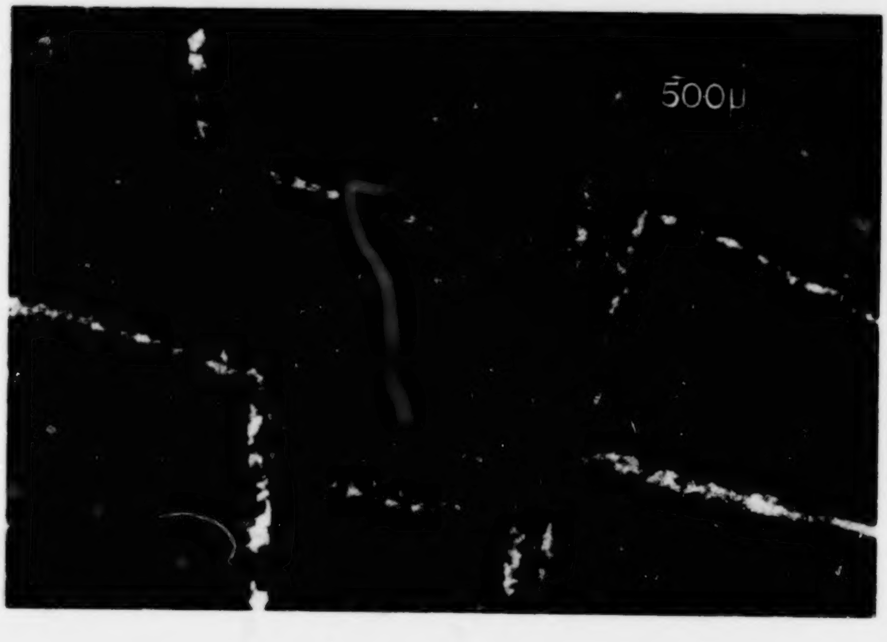


Photo 121. 50X (Transmitted Light- Crossed Nicols)

Offsets of various fractures within quartz-feldspar-sericite-chlorite groundmass. Fractures filled by illite-sericite ± quartz ± pyrite ± Fe hydroxides.

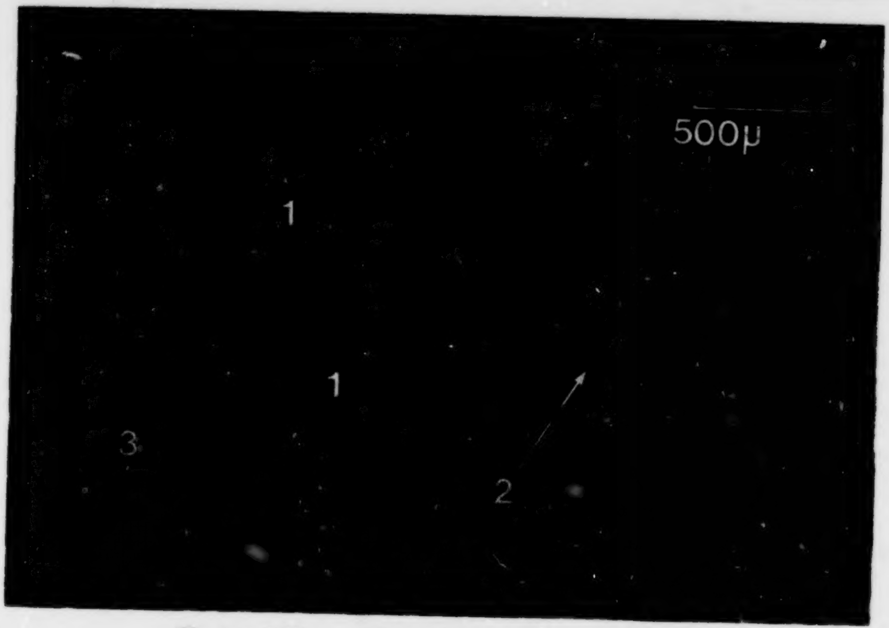


Photo 122. 50X (Transmitted Light)

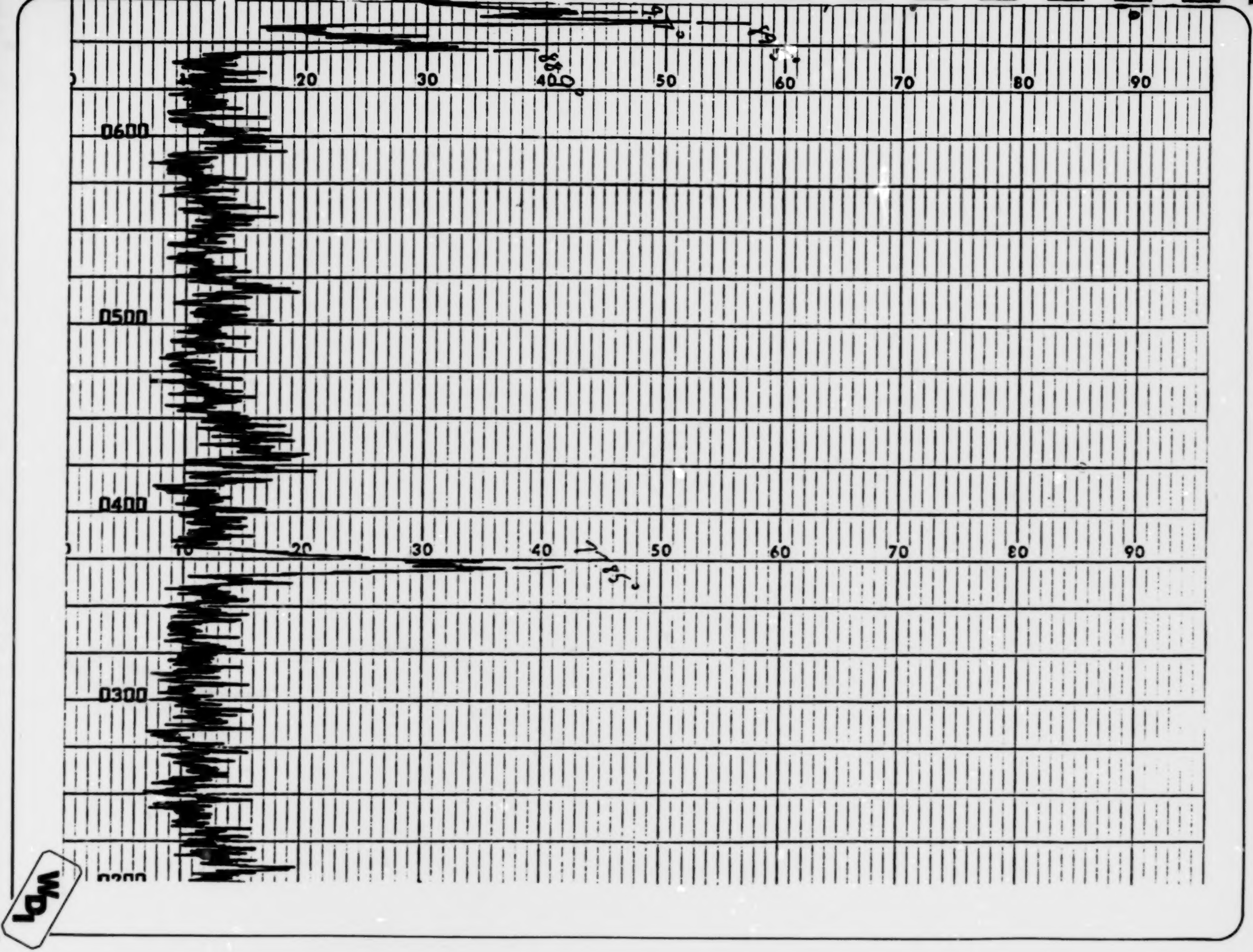
Breccia zone of quartzofeldspathic fragments (1) within matrix of illite-sericite-Fe hydroxides (2)-chlorite (3).

Photo 123. 50X (Transmitted Light - Crossed Nicols)

Breccia zone of quartzose fragments (1) in matrix of sericite-illite (2) within quartzofeldspathic groundmass (3).

APPENDIX III

X-Ray Diffraction Patterns



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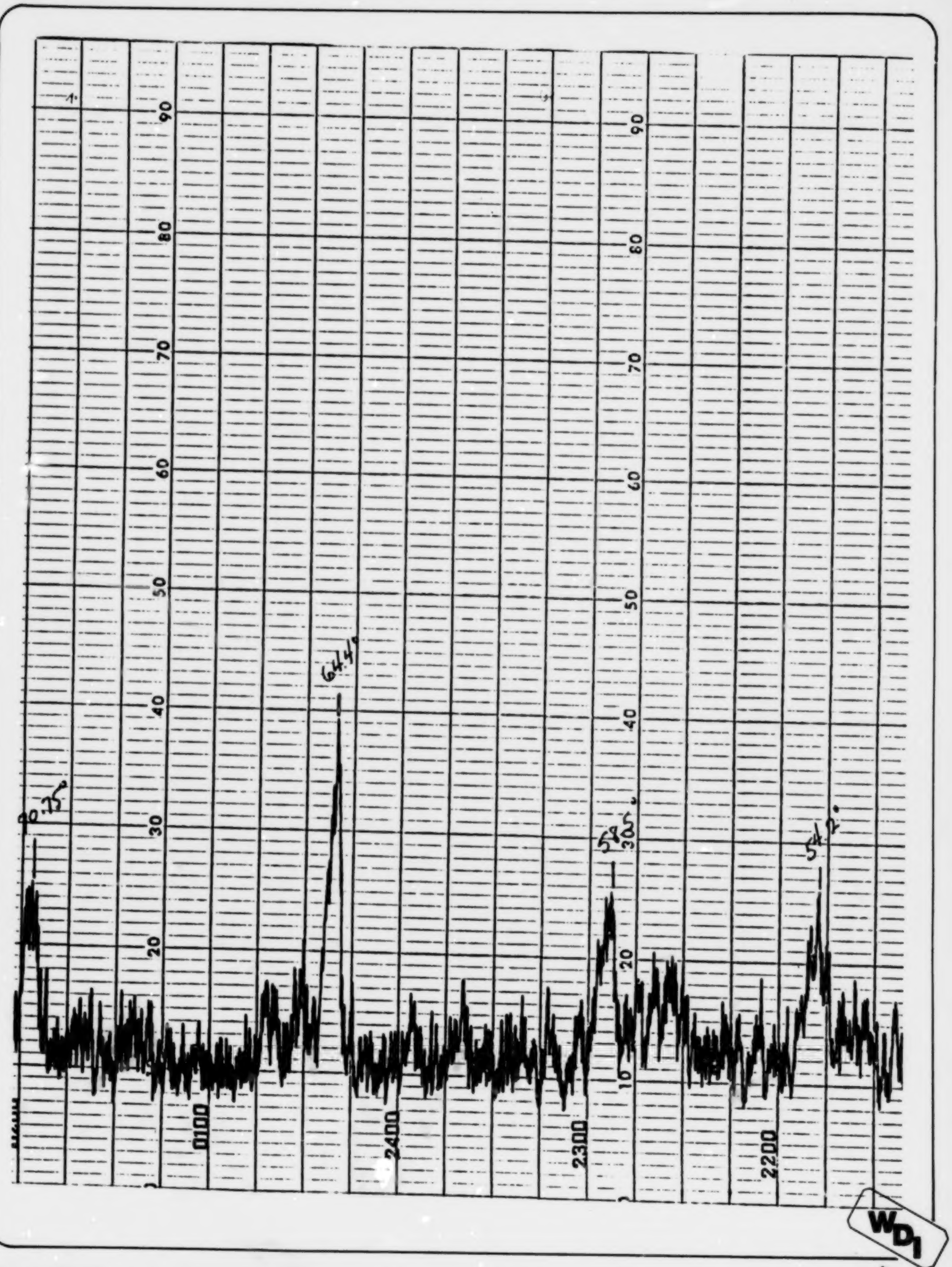
WD1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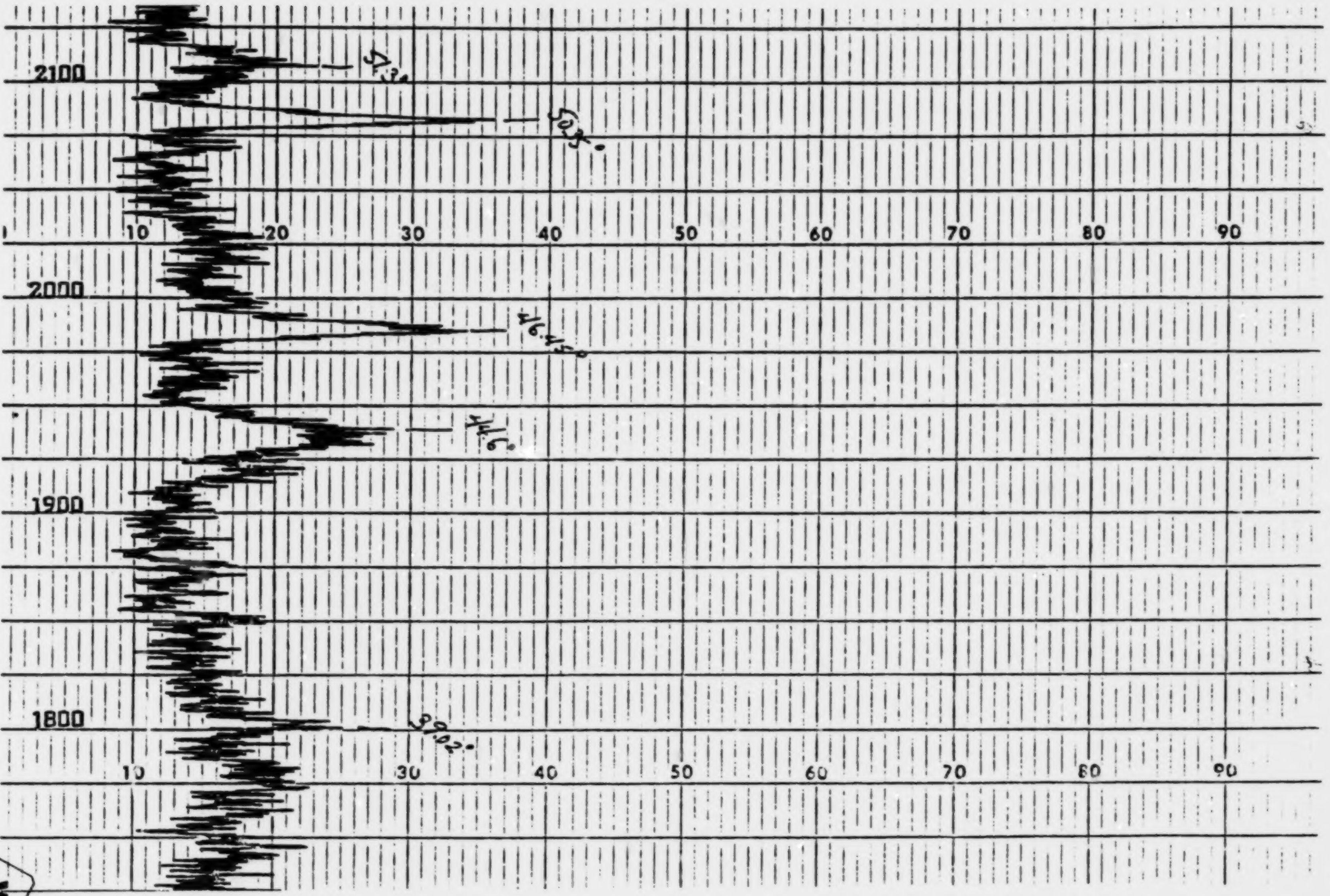
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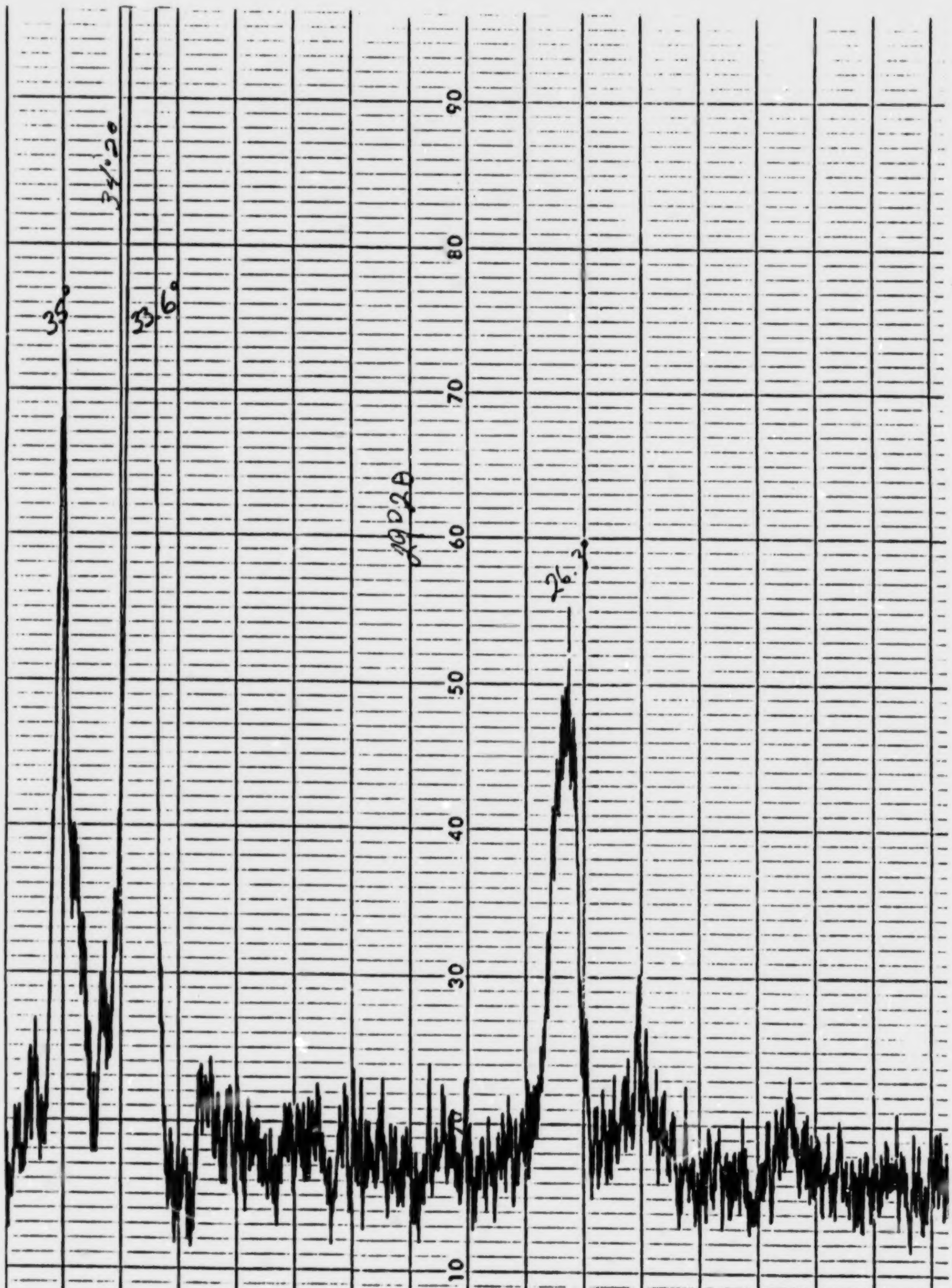
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26.3°

Volume

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WITTECK DEVELOPMENT INC.
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DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY

PHASE III

CAPITAL AND OPERATING COST
FEASIBILITY STUDY

Witteck Project 5209

for

Department of Supply and Services
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PHASE III

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1.0 SUMMARY

This Phase III report presents the capital cost, operating cost, and preliminary feasibility study for the Dawson Range heap leach project, more specifically known as the Antoniuk Deposit. This Phase III report is a continuation of the Phase I and Phase II parts of the study and uses the information developed in these earlier studies to develop the costs associated with bringing the property into commercial production.

Phase I of the study involved the evaluation of a number of samples from the Antoniuk deposit for their amenability for gold and silver recovery.

Phase II of the study involved a more detailed metallurgical and mineralogical evaluation of samples from the deposit, after an on-site visit to more closely study the problems involved. The Phase II study was reported in two volumes entitled "A Metallurgical Assessment" and "Mineralogy and Dye Penetration Testwork".

In this Phase III of the report, the capital cost, operating cost and a preliminary feasibility study was completed. In order to bring the Antoniuk deposit into operation at a mining rate of 743,200 tonnes per year, the capital cost would be Canadian $2.7 million. Using heap leach techniques to treat the ore on a 180 day per year leaching schedule to recover 75% of the gold content at a gold price of U.S. $400 per ounce would not be economically attractive. The gold recovered from the operation would be sufficient to cover the operating costs at a gold price of U.S. $450 per ounce. Operating costs at the property were estimated at Canadian $14.14 per tonne of ore treated. This high operating cost was the result of high cyanide consumptions during the leaching process, high ore haulage cost because of the heap leach location, and the relatively small scale of the operation. Reductions in cyanide consumption may be realized by the increased use of lime during the leaching process and deserves further investigation. Sensitivity analysis indicates that the operation can break even at an operating cost of between Canadian $10 and $12 per tonne of ore leached.

The ore grade of 0.99 grams per tonne during the first 3.5 years of the heap leach operation was another major factor in the economics of the proposed operation. Sensitivity analysis shows a head grade of between 1.4 and 1.5 g/tonne is required during the first 3.5 years of operation to achieve a break even point. High grade sulphide ore is available during the 3.5 to 5 year life of the operation at this higher grade, but also incurs a major increase in the ore to waste ratio which makes this phase of the operation less economically attractive.

The gold price has a major influence on the economics of the project and, at a gold price slightly above U.S. $500 per ounce, the Antoniuk heap leach project is at a break even operating cost.

A more detailed financial evaluation of the project should be undertaken to investigate the interactive effects of gold price, operating cost, head grade and gold recovery. A combination of all these factors in a positive direction to improve the project economics could result in the project being economically viable.

Pilot plant column testwork has indicated that the 75% recovery used in developing the economics may be a conservataive estimate of the actual heap leach recovery. Plant practice on test heaps and full scale heaps has also indicated that pilot plant column testing realize recoveries that are in the order of 2-5% lower than is expected in these larger heaps. Larger scale testing could verify the recovery in addition to providing the required evaluation of the head grade of a larger bulk sample.

2.0 INTRODUCTION

In the first two phases of the metallurgical and engineering assessment of the Dawson Range heap leach project, six surface bulk samples were evaluated for their potential for the recovery of their gold and silver content. The initial evaluation included mineralogical evaluation, permeability testing, bench scale agitated cyanidation leaches and pilot scale column heap leach testing. The results of the initial testwork on two of the samples from the Antoniuk deposit were very encouraging and resulted in a decision to proceed to a more detailed study investigating the engineering and economic feasibility of bringing the property into production using low cost heap leaching techniques.

This report presents costs for a 743,200 tonnes per year operation for the recovery of gold using cyanide heap leaching technology for the Antoniuk deposit. The heap leach is designed to handle 743,200 tonnes of gold ore per year for the five year life of the mine. The process plant and solution ponds are designed for processing of heaps up to 3,716,000 tonnes, which is the final size of the heap leach operation.

The proposed facilities include the following:

- Open pit mine (mined by a contractor)
- Two-stage portable crushing plant (possibly owned and operated by a contractor)
- A "valley type" heap leach pad
- 94,670 cubic meter pregnant solution pond
- 16,200 cubic meter barren solution pond
- 3,348 cubic meter impound pond

- 3,348 cubic meter sludge impound pond
- Zinc precipitation recovery plant
- Offices, refinery and change house
- Guard house
- Laboratory and laboratory equipment.

The report includes a metallurgical process description, process design criteria, plant description, capital cost estimate, operating cost estimates, schedule, preliminary equipment ledger, equipment and installation quotations, process flowsheets and preliminary layout drawings.

3.0 METALLURGICAL PROCESS DISCUSSION

3.1 Metallurgical Testwork

Metallurgical testwork was undertaken for the Antoniuk deposit by Witteck Development in two phases. The first phase consisted of testwork performed on samples from surface trenches in the deposit and were designated TR20 and TR24. The samples were subjected to agitated leach tests to determine the optimum leaching conditions. These agitated leach tests, which are the indication of the ultimate possible gold extraction, showed recoveries of 99% and 88% respectively on samples TR20N and TR24N. Large scale pilot plant tests on the one inch mill ore material confirmed that high recoveries (+95%) of gold were possible using the heap leach technique on the TR20 sample. However, results on the TR24 sample showed extractions between 38% and 50% and that further metallurgical work was required to determine the reason for these lower recoveries.

In Phase II testwork, the sample T21N from the Antoniuk deposit was designated for testing as it was the sample believed to be most representative of the deposit. However, the sample may not be representative of the entire deposit as it is not located in a section of the better mineralized zones of the deposit. In addition to testwork completed on this sample, testing was also carried out on the diamond drill core from the sulphide ore in the deposit. Detailed results of the testwork are presented in Volume I of the report. Summarizing the important factors from that report, the column heap leaching tests indicated the following:

1. The final extraction of gold from Sample TR21N was independent of the feed size of the material. However, longer leach times of up to 90 days were required for coarser material to attain the final level of extraction.

2. Gold extractions varied from 68% to 87% for the Antoniuk TR21N samples tested.

3. Cyanide consumptions ranged from 3.4 to 11.3 kg/t for the TR21N sample. Higher lime additions of 3.9 kg/tonne of ore indicated a reduction in cyanide consumption to a more economical 2.0 kg/t for a sample of material washed to passing 1 1/2".

4. Gold extractions on the drill core samples varied from 28% to 60% and with cyanide consumptions in the order of 3.5 to 5.0 kg/ton.

5. The average head of the Antoniuk ore tested was 0.77 g/t. This was significantly lower than the average of the ore reserves delineated by subsequent diamond drilling which averaged 0.99 g/t. Although special care was taken to minimize the "nugget" effect during sampling procedures, deviations in head assays were evident. For metallurgical evaluation purposes, recoveries were calculated on average head assays and residue assays for each test.

3.2 Mineralogical Description

The Antoniuk deposit can be described as a porphyry gold deposit. It comprises a heterolithic breccia body transected by rhyolite porphyry dykes and small scale structural dislocations. The breccia zone exhibits various degrees of fracture due to faults, shears and secondary infilling by replacement minerals.

Mineralization of the Antoniuk breccia comprised of slightly argentiferous gold occurring as coarse free gold. The gold is locally associated as inclusions within secondary vein system after the sulphide mineralization. Many additional minerals characteristic of epithermal-porphyry systems were observed during the mineralogical examination.

The dye penetration testwork indicated weak to excellent fracture controlled permeability, depending upon the sample under investigation. The degree and type of permeability was highly variable from sample to sample. This variability was controlled to a major extent by the degree of silicification, sericitization and oxidation and the degree of brecciation and hydrofracturing of the sample.

The gold principally occurred as coarse 50 to 300 micron free particles, but with lesser associates with iron oxyhydroxides, silicates and sulphides. The presence of base metal sulphides including sphalerite, chalcopyrite and molybdenite may cause increased cyanide consumption, and treatment of the solution recycled to the heaps and the final effluent to prevent a build-up of deleterious metals in the system may be required.

3.3 Open Pit Mining

Open pit mining will be contracted to a mining contractor experienced in the type of mining and familiar with the local weather conditions and environment. The contractor will supply all the necessary mining equipment, maintenance facilities and camp facilities to support his operation. Provision has been made in the plant design to provide suitable areas for his garage and the service facilities necessary for his operation.

A mining consultant will be used for the development and layout of the initial pit design.

The ore will be trucked via a 2.5 km haul road from the open pit mine to the plant site. At the plant site, it will be directed either to the 10,000 tonne coarse ore stockpile or directly to the crushing plant for crushing to minus 35 mm.

Ore reserves at the Antoniuk were calculated under the direction of Archer Cathro. An independent consultant. E. Holt, was retained to calculate ore reserves and the results are shown in Table 1.

A subsequent review of the correlation between the drilling and trench sample results showed that the grade of the deposit could have been as high as 1.14 g/t. However, for purposes of this study, the head grade was assumed to be 0.99 g/t.

3.4 Ore Handling and Crushing

Ore from the mine is dumped directly from the haulage truck or from the loader at the stockpile over the grizzly and into the 9 cubic meter capacity feed hopper. Ore is withdrawn from the hopper into the 1000 mm x 625 mm chassis mounted portable jaw crusher. Crushed product is conveyed to a surge bin, where a feeder controls the feed rate to the 4.25 ft cone crusher which is set at 20 mm to give a product size of 100% passing 35 mm. This final crushed product is conveyed to a truck loading bin, whence it is hauled to the heap.

To reduce capital costs of this crushing system, a used portable crushing plant of this capacity can be obtained at the time of writing for approximately $500,000. The mining contractor has quoted an attractive price of $1.85 per tonne of material crushed. This figure has been used in preparation of the operating cost estimate.

3.5 Heap Leaching

Heap leaching is a simple, low cost method of recovering gold and silver where grade or reserves are not sufficient to

Table 1

ANTONIUK DEPOSIT - PERMIAN JOINT VENTURE

ORE RESERVES

(Calculated by E. Holt, P.Eng.)

METRIC UNITS

Category	Ore (Tonne)	Grade (gm/tonne)	Cumulative (grams)	Waste (Tonnes)	Waste /Ore
0.5 gram/tonne cut off					
Oxide	2,622,000	0.99	2,596,000	741,000	0.28
Sulphide	1,094,000	1.50	1,641,000	2,405,000	2.20
Combined	3,716,000	1.14	4,236,000	3,147,000	0.85
0.7 gram/tonne cut off					
Oxide	1,892,000	1.14	2,257,000	1,470,000	0.78
Sulphide	1,069,000	1.52	1,625,000	2,430,000	2.28
Combined	2,961,000	1.28	3,790,000	3,901,000	1.32

justify the capital or operating expenses of a conventional or carbon-in-pulp plant. Heap leaching is particularly suitable to low cost open pit mining of ore containing 1-2 grams Au per tonne or 80 to 100 grams Ag per tonne. The Antoniuk ore body may be developed using heap leaching technology because it meets this qualifications.

Simply stated, heap leaching is a process whereby broken or crushed gold/silver ore is stacked on an impervious pad. Dilute sodium cyanide solution (0.05 to 0.10% cyanide) is applied to the top of the heap. As the solution percolates through the heap, the finely disseminated precious metals are dissolved. The pregnant solution, enriched with gold is collected from the base of the heap through a system of channels into the pregnant solution pond. From the pond, the pregnant solution is pumped to a process plant where the precious metals are removed from the solution. The process plant is discussed in Section 3.6 of this report. The barren solution is then recirculated back into the barren solution pond, before it is pumped to the heap for further leaching. The process is a closed circuit.

The concept of heap leaching dates back to about 1750, when acid solutions were percolated through large heaps of copper oxide ore on the banks of the Rio Tinto in Spain. Since then, the process has been used extensively throughout the world, largely to leach copper and uranium ores. Cyanide heap leaching of gold ores, however, is comparatively new. This concept was first developed by the United States Bureau of Mines in the mid 1960's. Since that time, heap leaching of gold and silver has become a viable process for gold recovery in arid, semi-arid and tropical climates for many years. Several references have been published describing the technology. In Canada, however, heap leaching technology has met several stumbling blocks. The two largest stumbling blocks slowing the utilization of technology include:

1. Adverse climatic conditions
2. Physical characteristics of the ore.

In 1980, the Ontario Ministry of Natural Resources investigated heap leaching technology with respect to its feasibility in Canada on Archean gold ores (Ontario Geological Survey Open File Report 5258). The results of their studies were encouraging and it was felt that the technology could easily be modified to suit the Canadian Mining Industry. As well as leaching high tonnages of very low grade ore, the technology can be utilized for very small tonnages of high grade ore. Low initial capital and lower operating costs provide an excellent situation for the small mine operator to commence operations on a property and produce cash flow for further expansion.

Additional studies and reports have been published which clearly indicate the feasibility of economically utilizing heap leach technology for the development of Canadian gold deposits.

The Antoniuk heap leach is designed to handle the total of 3,716,000 tonnes of ore presently outlined. The heap has been designed to use one of the valleys in an area in close proximity to the plant site. A starter dam of imported material is first constructed, and the area upstream of the dam, including the dam face, is lined with the impermeable membrane. Ore is then stacked using this starter dam as the toe" of the heap. In successive years, the heap area is expanded and the impermeable membrane area expanded as required. One benefit of building the heap in lifts is the continued exposure of all the ore to the leaching solution. It is expected that this will provide an increase in the gold recovery for no additional operating cost. Tailings disposal costs are reduced because the leached ore will remain on the impermeable heap leach pad and will not require disposal to a suitable and environmentally safe area.

The solution ponds and process equipment are designed to treat the necessary solution flow for treating the final area of the heap leach pad.

3.6 Metallurgical Extraction

Two types of metallurgical extraction circuits are most commonly used for heap leaching. They are as follows:

1. Zinc precipitation
2. Carbon adsorption

Zinc precipitation can be used for recovery of gold and silver in solutions from heap leach operations. Solution is pumped from the pregnant solution pond through a precoated clarifying filter to remove all suspended particles. The clarifying filter is generally precoated with diatomaceous earth or other suitable cake to provide a solution with less than 3 parts per million solids. The clarified solution is then de-aerated to approximately 0.5 to 1.0 ppm O_2 in a packed vacuum tower operating at minus 22 to minus 26 mm Hg vacuum. Zinc dust is added to the de-aerated solution in order to precipitate the gold and silver. The precipitate is collected on a filter and removed periodically for smelting to Doré metal.

Carbon adsorption can be used for recovery of gold from leach solutions, but not for recovery of silver because of the latter's poor adsorption/desorption characteristics on activated carbon.

The zinc precipitation process has been selected for the Antoniuk project because of its metallurgical suitability and lower cost for the size and scope of overall project. Activated carbon requires reactivation in a high temperature kiln and acid treatment before being reintroduced in the adsorption system. The reactivation and treatment equipment is very expensive and requires experienced operators. Fewer operators are required with less technical ability for operation of a zinc precipitation circuit than with a carbon adsorption circuit. Zinc precipitation is a well-known technology in Canada, and thus it is easier to find an experienced operator capable of solving operational problems. The equipment is also very common throughout the mining industry and asset recovery will be must easier than with custom-built carbon adsorption equipment.

3.7 Gold Refining

The gold bearing zinc precipitate must be stripped from the filters and smelted for the production of Doré metal. The precipitate is blended with refinery reagents and slag from previous pours to provide the proper fluxing of the molten metals.

The most common type of bullion furnace is the direct fired pot type furnace. The furnace shell, lined with high temperature refractory, is mounted on trunnions that allow for gentle rocking and pouring during the smelting operation. Compressed air and fuel oil are generally injected by a burner located next to a port on the rotating axis of the furnace.

The zinc precipitate is thoroughly blended with borax, silica flour, fluorspar, soda, nitre and manganese dioxide before being charged into the preheated furnace.

Additional slag may be charged into the furnace in order to collect any entrapped gold and to assist in providing a more fluid pour. After approximately five or six hours of furnacing, the Doré metal is poured into molds and removed for sale.

The refinery for the property will be located within the process building. Two Wabi type furnaces are provided for smelting the precipitate produced from the Merrill-Crowe precipitation process. A vault is provided for short term storage of precipitate and bullion.

4.0 PROCESS DESIGN CRITERIA

4.1 Introduction

The basic concept of heap leaching is very simple. However, as with any production operation, clear and concise design criteria must be established prior to design, construction and operation of the Antoniuk property.

Since the ore grade and reserves have been determined under the supervision of Archer Cathro and the location of the ponds, heaps and process plant has been established, four major items must be considered. These items include the following:

1. Applicability of the ore to heap leaching
2. Location and climatic considerations
3. Production rate and operating schedule
4. The method of metallurgical extraction.

The general plant design criteria has been based on the following criteria, using a cut off grade of 0.5 grams per tonne:

1. The mining and leaching of 743,200 tonnes per year of oxide ore assaying 0.99 grams per tonne of Au for the first 3.5 years of mine life with an extraction of 70% of the gold content.

2. The mining and leaching of 743,200 tonnes per year of sulphide ore during the ensuing 1.5 years of mine life. This material assays 1.5 g/tonne but recoveries of 60% are anticipated.

The process design criteria were compiled to permit preparation of the feasibility study, process flow diagram and preliminary layout drawings. The design criteria were based on testwork conducted at Witteck Development Inc., as discussed in Section 3.1 of this report and data from Archer Cathro reports. Where design data were not developed during this metallurgical investigation, information has been utilized from existing or similar installations.

In some cases, where reasonably comparable information was not found, the design criteria were based on sound engineering principals, judgements and experience.

4.2 Applicability

Heap leaching has been applied with remarkable success to a wide variety of gold and silver ores where grade or reserves are not sufficient to justify the capital and operating expenses of a conventional carbon-in-pulp plant. Heap leaching is particularly suitable to low cost open pit mining or to waste rock containing 2 grams Au per tonne or 70 to 100 grams Ag per tonne.

Heap leaching is less likely to be applied to ore from a deep underground mine. If the ore can justify mining costs of $20.00 to $55.00 per tonne in the first place, it will probably justify the higher capital and operating cost of $5.00 to $10.00 per tonne of a conventional mill in order to achieve a higher recovery.

The applicability of the Antoniuk ore deposit depends on the following criteria:

1. The permeability of the solution through the leach must be considered in order to determine the degree of crushing and the associated cost to obtain economical recovery. The Antoniuk ore should be crushed to minus 35 mm in size based on testwork conducted to date and as discussed in Section 3.1 of this report. Additional testing may indicate that less crushing is required and therefore the anticipated operating costs may be reduced.

2. The reagent consumptions of cyanide and lime must be sufficiently low to provide economical operation and recovery. The cyanide and lime consumption has been assumed as follows, based on the laboratory testwork:

   - cyanide : 2.0 kg per tonne of ore
   - lime : 4.0 kg per tonne of ore.

3. Sufficient days must be available in the year to allow heap leaching to continue to operate. The ore will be leached will be for 90 days and leaching will terminate before the average daily temperature drops below 0^oC. The period available to continue operation on a commercial scale is discussed in Section 4.4 of this report, but is scheduled for 180 days per year based on climatic data for Carmacks, Yukon.

4. The permeability of the ore heap determines the method of solution distribution, the height of the piles and the method of mining and handling ore to the heaps. These considerations are described in Section 5.0 of this report.

4.3 Project Location

The Antoniuk property is located within a 150 km long linear zone known as the Dawson Range Gold Belt, which is part of the Dawson Range in the west central Yukon. The property location is in gently rolling country with well developed valleys draining the area. These valleys are separated by rounded broad ridges which have relatively sparse surface vegetation and well spaced coniferous trees. The valleys are relatively well wooded with coniferous trees being predominant. The area is well drained with several creeks and gulches cutting across the property.

The Antoniuk property lies near the south-east end of the Dawson Range Gold Belt. It is accessible by road. The nearest community is Carmacks which is located 60 km south-east of the property. The nearest major centre is Whitehorse, which is a further 160 km south of Carmacks by paved highway.

Overburden on the Antoniuk property is thin and ranges from one metre in thickness on steep south-facing slopes to ten metres on north facing slopes. There is evidence of permafrost throughout the deposit and surrounding area.

4.4 Project Climatic Conditions

The climatic conditions that have the greatest significance to the design, construction and operation of a heap leach in the Yukon are as follows:

1. Low temperatures
2. Wind
3. Precipitation (rain, snow, fog, sleet, etc.)

Heap leaching cannot be undertaken during the extreme cold weather periods of the year because the cyanide solution freezes within the heap, in pumps, tanks, pipes, ponds and any other equipment exposed to the elements. New techniques have been developed, however, that have allowed some operations to continue even though atmospheric temperatures drop as low as -20°C.

The following data is used to evaluate the effects of climate on heap leaching operations:

1. Maximum Mean Temperature per Month
   This value is established by determining the average maximum temperature on a daily basis.

2. Minimum Mean Temperature per Month
   This value is established by determining the average minimum temperature on a daily basis.

3. Mean Temperature per Month
   This value is established by determining the average temperature on a daily basis.

4. Heating Degree Days
   This value is determined from the average daily temperature, less 20°C, multiplied by the number of days in the month. This value is used on a monthly basis for comparative analysis and any comfort heating evaluations with the solution degree days.

5. Solution Degree Days

This value is established from the average daily temperature less 0°C multiplied by the number of days in the month. Since cyanide solutions used for heap leaching are very dilute, the freezing temperature of the solution is 0°C. The solution degree days is used to determine the freeze/thaw index and determine the number of operating days for the heap leach on a yearly basis.

6. Wind

Wind affects both freezing and thawing, increased evaporation rate of the cyanide solution from the heaps and ponds and can cause leach solution losses as tiny droplets in wind-blown mist. Consequently the effect of wind is considered during design of the leach pile and the selection of the solution distribution system. Wind may also create a problem with respect to drifting of snow on or around the heap.

7. Precipitation

Precipitation can create a variety of problems for heap leach operations. Rain dilutes the concentration of cyanide and lowers the pH of the solution. The ponds must be sized to handle leach solution, rain and melted snow collected from the heap, ditches and the ponds themselves.

The freeze/thaw data and the number of heap leaching days is presented in Table 2 and Figure 1. Precipitation data is presented in Table 4 and Figures 3 and 4. The temperature data for Carmacks, Yukon is presented in Table 3 and Figure 2.

Examination of climatic data for Carmacks, Yukon, indicates that the daily mean temperature averages above 0°C for six months of the year, April through September. Of these months, the average daily minimum temperature dips below 0°C in October and May (-6°C and -.2°C respectively). The calculated thawing/freezing index indicates a net thaw for April and May, which will allow operations to begin so long as solutions are kept moving when temperatures drop below 0°C. As indicated in the weather data, heap leaching may be possible for 214 days of the year.

Experience at heap leaching operations in Nevada and Canada however, has shown that operation can continue through winter, even at temperatures below -20°C. Snow and ice build up on the piles effectively insulating the pile, allowing the solutions to percolate through. Leach solutions penetrate the snow and ice as long the sprays are kept operating. Alternatively, leach solution has been added through inverted drums set in small ponds on tope of the pile; solution flows beneath the ice and snow layer formed on the top of the heap ponds. Direct injection of leach solutions into tall or valley heaps in order to extend the leaching season into the winter months is another method.

It is therefore proposed that for commercial operation, the operating season at Antoniuk for seven months from April through October should allow two 90 day leaching campaigns per year. During periods of low temperature, solution will either be kept moving or lines will be drained. Once shutdown for the winter season, however, leaching operations are not expected to resume until mid-April or May or when outside temperatures permit.

Senninger Wobblers have been selected for solution distribution as they provide good dispersion in relatively large droplets. Direct injection of leach solution may be considered in order to extend the leaching season into November if required.

Table 2

CLIMATOLOGICAL DATA FOR CARMACKS, YUKON
NUMBER OF HEAP LEACHING DAYS AND FREEZE/THAW INDEX

Month	Days per month	Freeze/Thaw Index	Available Leaching Days
Jan.	31	9	0
Feb.	28	7	0
Mar.	31	5	0
Apr.	30	0	30
May	31	0	31
June	30	0	30
July	31	0	31
Aug.	31	0	31
Sept.	30	0	30
Oct.	31	0	31
Nov.	30	4	0
Dec.	31	7	0
Total	365		214

Figure 1

YUKON – HEAP LEACH FREEZE/THAW DATA

TIME(MONTHS) VS. FREEZE/THAW INDEX

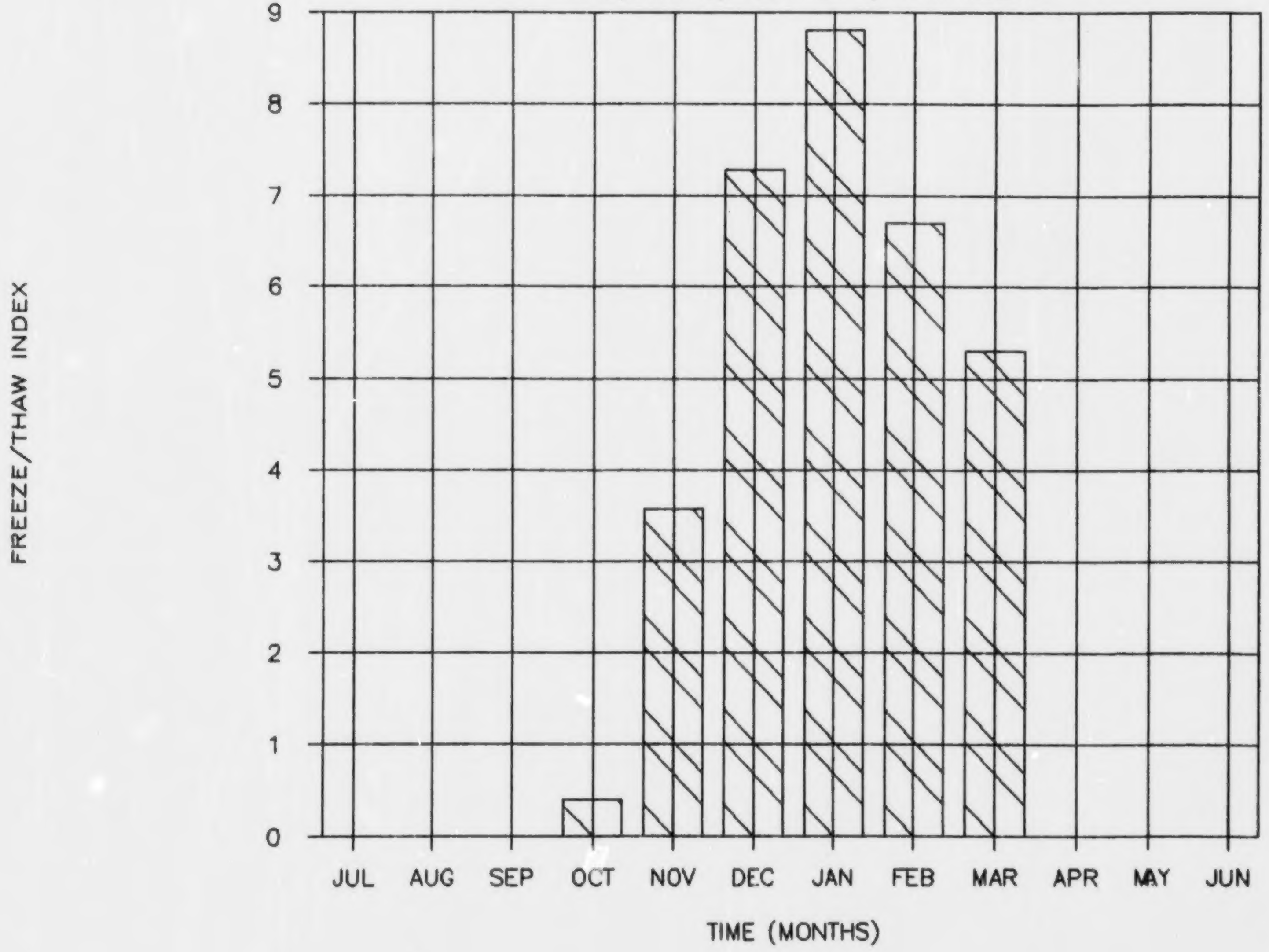


Table 3

CLIMATOLOGICAL DATA FOR CARMACKS, YUKON

TEMPERATURE DATA BY MONTH

Month	Days per month	Max.Temp. (C)	Min Temp. (C)	Average Temp. (C)	Heating Degree Days	Solution Degree Days
Jan.	31	-24.1	-33.9	-29.0	1457	899
Feb.	28	-11.9	-24.9	-18.4	1019	515
Mar.	31	- 2.0	-18.6	-10.3	877	319
Apr.	30	8.1	- 6.5	0.3	531	0
May	31	14.4	- 0.2	7.1	338	0
June	30	20.3	5.3	12.8	156	0
July	31	21.8	7.6	14.7	102	0
Aug.	31	19.5	5.1	12.3	177	0
Sept.	30	13.2	0.2	6.7	339	0
Oct.	31	3.4	- 6.1	- 1.4	600	42
Nov.	30	-10.2	-18.5	-14.4	971	431
Dec.	31	-20.0	-29.5	-24.8	1325	767
Total					7892	2973
Average		2.6	-10.0	- 3.7		

Figure 2

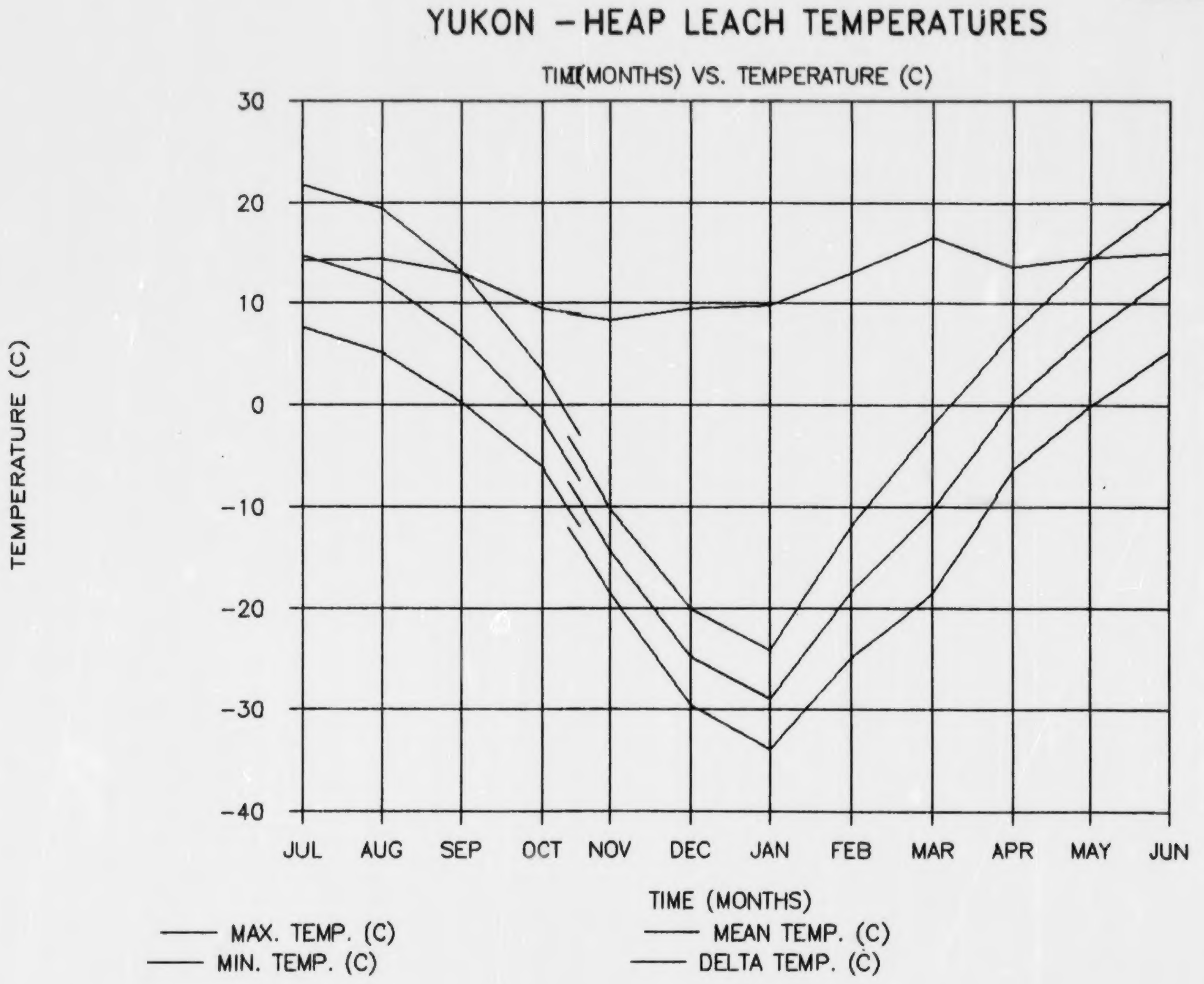

YUKON -HEAP LEACH TEMPERATURES
TIMI(MONTHS) VS. TEMPERATURE (C)
30
20
10
0
-10
-20
-30
-40
TEMPERATURE (C)
JUL
AUG
SEP
OCT
NOV
DEC
JAN
FEB
MAR
APR
MAY
JUN
TIME (MONTHS)
MAX. TEMP. (C)
MIN. TEMP. (C)
MEAN TEMP. (C)
DELTA TEMP. (C)

Figure 3

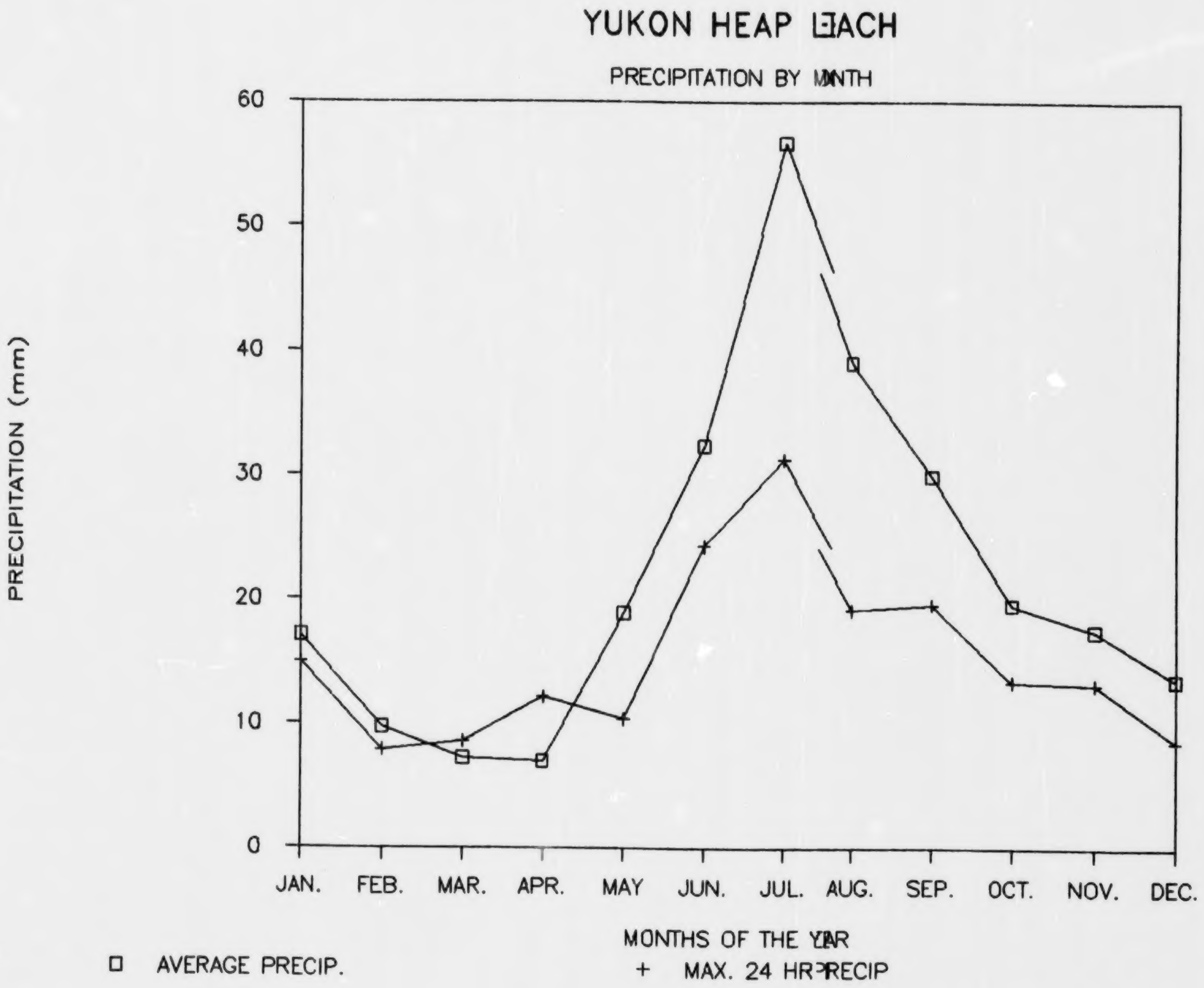
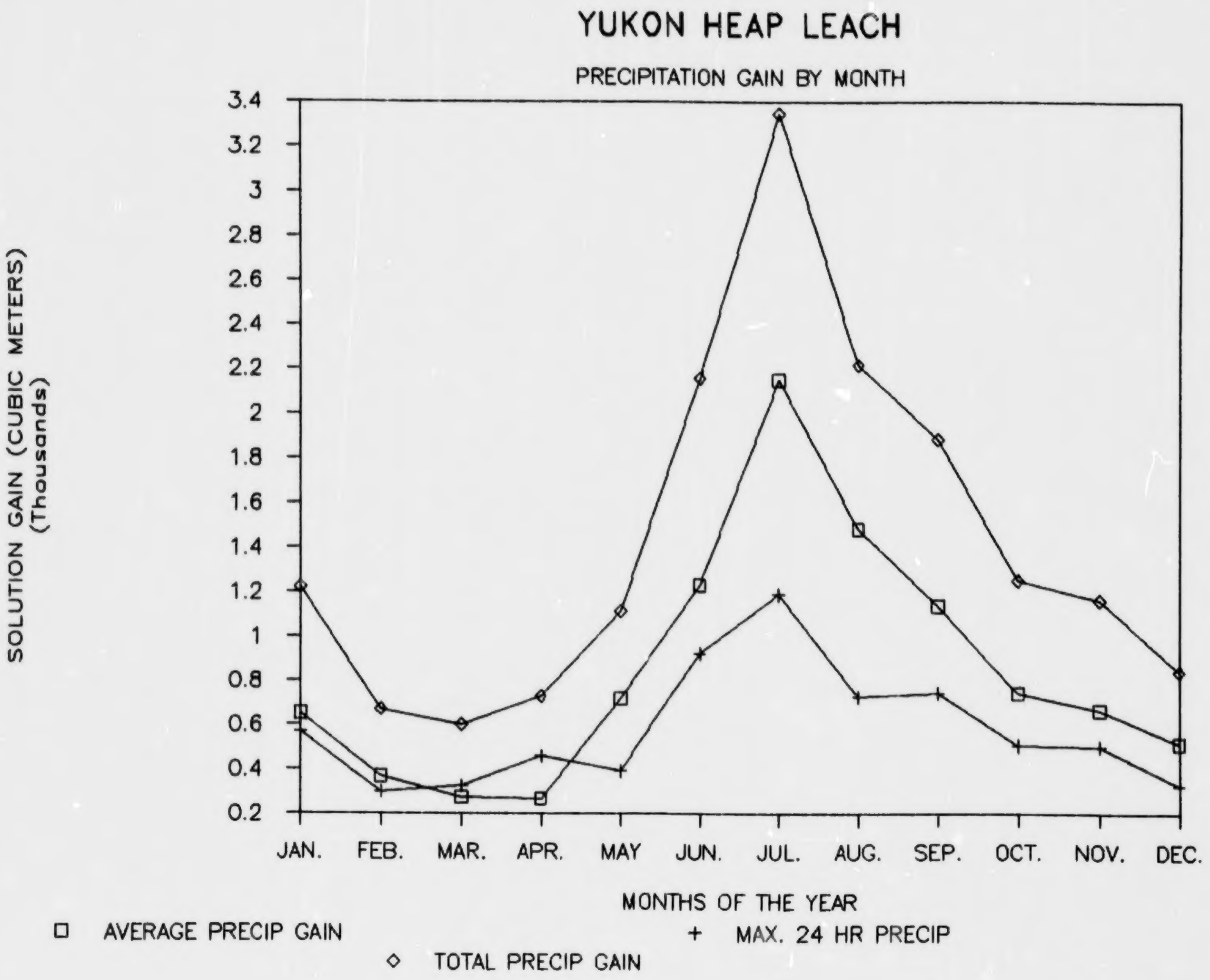

YUKON HEAP LEACH
PRECIPITATION BY MONTH
PRECIPITATION (mm)
60
50
40
30
20
10
0
JAN.
FEB.
MAR.
APR.
MAY
JUN.
JUL.
AUG.
SEP.
OCT.
NOV.
DEC.
MONTHS OF THE YEAR
□ AVERAGE PRECIP.
+ MAX. 24 HR PRECIP

Table 4

CLIMATOLOGICAL DATA FOR CARMACKS, YUKON
PRECIPITATION BY MONTH

Month	Days per Days per Month	Total Precipitation (mm)	Greatest Precipitation in 24 hrs (mm)
Jan.	31	17.12	15.0
Feb.	28	9.75	7.9
Mar.	31	7.26	8.6
Apr.	30	7.00	12.2
May	31	18.96	10.4
June	30	32.46	24.4
July	31	56.69	31.4
Aug.	31	39.17	19.2
Sept.	30	30.00	19.6
Oct.	31	19.59	13.4
Nov.	30	17.50	13.2
Dec.	31	13.61	8.6
Total		269.11	
Average		22.43	15.33
Maximum		56.69	31.4
Minimum		7.00	7.9

Figure 4


YUKON HEAP LEACH
PRECIPITATION GAIN BY MONTH
SOLUTION GAIN (CUBIC METERS)
(Thousands)
3.4
3.2
3
2.8
2.6
2.4
2.2
2
1.8
1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
JAN.
FEB.
MAR.
APR.
MAY
JUN.
JUL.
AUG.
SEP.
OCT.
NOV.
DEC.
MONTHS OF THE YEAR
□ AVERAGE PRECIP GAIN
+ MAX. 24 HR PRECIP
◇ TOTAL PRECIP GAIN

4.5 Mine Design Criteria

Ore reserves	- oxide	2,622,000 tonnes
	- sulphide	1,094,000 tonnes
	- combined	3,716,000 tonnes
Ore grade	- oxide	0.99 grams/ton
	- sulphide	1.50 grams/ton
	- combined	1.14 grams/ton
Waste/ore ratio	- oxide	0.28/1
	- sulphide	2.20/1
	- combined	0.85/1
Mine life	- oxide	3.5 years
	- sulphide	1.5 years
	- combined	5.0 years
Mining rate		743,200 tonnes per year
Mine operations		300 days per year
Crushing plant	- operating weeks per year	40
	- operating days per week	6
	- availability	75%
	- capacity	172 tonnes per hour
	- maximum feed size	600 mm
	- product size	100% passing 35 mm

4.6 Metallurgical Design Criteria

Total heap tonnage	3,716,000 tons
Tonnage treated by heap leach per year	748,300 tons
Average crushing plant tonnage	172 tonnes per hour
Bulk specific weight of crushed ore	1600 kg/M^3
Angle of repose of ore	37^{o}
Heap percolation rate	0.25 litres/min/m^2
Pregnant pond capacity	94670 m^3
Barren feed capacity	16,200 m^3
Solution impound pond capacity	3,348 m^3
Sludge impound pond capacity	3,348 m^3
Leaching days per year	214 days
Leaching season	April to October
Cyanide consumption	2.0 kg/tonne ore
Lime consumption	4.0 kg/tonne ore
Zinc dust consumption	7 g/tonne ore
Diatomaceous earth consumption	8 g/tonne ore

4.7 Nomenclature

Metric (SI) units are used for the design criteria, design drawings and the feasibility study.

Quantity	Unit	Symbol
Length	metre	m
Mass	kilogram	kg
Time	second	s
Electric Current	ampere	A
Electric Power	kilowatt	kW
Temperature	degrees centigrade	C
Angle	degrees	°
Area	square metre	m^2
Volume	cubic metre	m^3
Frequency	bertz	HZ = cycle/sec
Density	kilogram per cubic metre	kg/m^3
Energy	joules	J
Concentration	grams per tonne	g/tonne

Other abbreviations used throughout this document are presented below.

>	greater than
≥	greater or equal to
<	less than
≤	less than or equal to
–	minus
O/F	overflow
O'size	over size material
U'size	under size material
%	per cent
+	plus
HDPE	high density polyethylene
PVC	polyvinyl chloride
U/F	underflow
ppm	parts per million (weight basis)

5.0 PLANT DESCRIPTION

5.1 Introduction

This plant description is provided as a general guide to the process operations, the process equipment and the basic premises for the design. The description is not intended to be a detailed technical representation of the process and support facilities. Some of the reasoning behind the design decisions are noted. General basis for some of the equipment selection and design, materials of construction, and environmental consideration as they relate to the process are included. The process flowsheets (Dwg. Nos. 10-110-001 and 002), site plan (Dwg. No. 10-130-001) and plant layout (Dwg. No. 10-120-001) are considered complementary to this description.

Area designations and numbers have been assigned to the various sections and unit processes within the limits of the project. The designations and area numbers have been assigned as follows:

Area No.	Designation
10	General
15	Mine/Pit
20	Crushing and Screening
30	Heap Leaching
40	Process Plant
41	Clarification
42	De-aeration
43	Zinc Precipitation
44	Reagent Storage
45	Reagent Mixing
50	Refinery
55	Cyanide Recovery
60	Services

All equipment is labelled and the Equipment Ledger is shown in Appendix I.

5.2 Area 10 - General

The General Area includes the total mining property encompassing all areas of the property.

A guardhouse consisting of a trailer building is included with roadwork, general civil site work, site drainage and a portable generating set.

The guardhouse is located on the road entering the mine property. All persons entering and leaving the property are required to sign in and out.

The portable generator set is rated at 500 kW to produce all 600 volt, 3 phase, 60HZ electrical power for the process plant and services for the property. The generator is self-contained in a closed van type trailer and is supplied complete with a diesel fuel tank, control panel and main circuit breaker. The power distribution to the property will be from the generator set. No provisions have been made for a back-up source of electrical power supply. No allowance has been made for a power source for the crushing plant. As described in Section 20, this will be included in the contract for the crushing of the heap material.

For the purposes of this evaluation, it has been assumed that no living facilities have been provided for the operation and that the work force will commute from Carmacks.

5.3 Area 15 - Mine/Pit

Open pit mining is performed by a contractor. After blasting, the ore is loaded by front end loaders on to trucks for haulage to the 10,000 tonne coarse ore stock pile, or directly to the crushing plant. All mining is accomplished by a contractor paid on a "per tonne" basis. The mined ore tonnage is determined by the belt weightometer located on the discharge conveyor from the crushing plant.

5.4 Area 20 -Crushing and Screening

All equipment including crushers, screens, bins and conveyors are installed in the crusher enclosure shown on the site plan. For the initial project design, a crushing plant owned and operated by the mine owner was envisaged. This provides the necessary monitoring and supervision of the mining contractor's performance and ensuring the required product size of the crushed ore. However, one potential contractor has offered an attractive price for crushing the ore, which would reduce the capital cost of the project.

The facility described below is workable and appropriate to producing 172 tonnes per hour of 100% minus 35 mm crushed product.

Coarse ore is transferred by a front end loader from the coarse ore stockpile or dumped directly from a haulage truck on to the 216 mm grizzly located over a 9 cubic meter coarse ore bin. Grizzly oversize is broken using the installed rock breaker.

Coarse ore is drawn from the surge bin by a feeder feeding a 1000 mm x 625 mm jaw crusher set at 50 mm. This crusher product discharges on to a 610 mm wide conveyor feeding a surge bin using a transfer conveyor.

Ore from the surge bin is withdrawn at a controlled rate via a feeder and is crushed in a 4.25 ft Symons cone crusher. The crusher product discharges on to the final product conveyor where lime is added to the belt to ensure intimate mixing of the lime with the ore.

The mixed ore is conveyed to the truck loading bin from which ore is trucked to the leach pad. The belt weightometer located on this belt monitors both the crushed ore for accounting purposes.

5.5 Area 30 - Heap Leaching

The heap leaching area includes the heap, pregnant, barren and sludge ponds and the process building. The pregnant solution pond is located within the leach pad itself. A dam is constructed across the gulch to the 970 foot elevation level, thus creating a pond. This pond area is specially prepared before being lined with the 60 mil thick high density polyethylene (HDPE) liner. The liner is then covered with sand and gravel to prevent damage to the liner during the heap loading cycle. The crushed ore is loaded into the lined area as shown in Drawing No. 10-110-001. Pregnant solution is contained behind the dam and within the voids in the ore, and is pumped out of the dam using two pregnant solution pumps installed in two wells inside the dam wall.

The pad supporting the heap is impermeable in order to prevent precious metal losses, as well as to prevent the egress of cyanide solutions that could leak into the water table. The liner construction consists of the following:

- 100 mm compacted sand
- 200 mm compacted clay
- 50 mm compacted sand
- 60 mil high density polyethylene
- 200 mm sand
- 250 mm gravel

The heap liner slope follows the natural slope of the ground on the up-stream side of the dam.

A test well is installed in the dam to test for any leakage of cyanide solutions from the pregnant solution pond.

The pregnant solution pond has a capacity of 94760 m^3 to a level below the top of the contained dam. The pond is sized to handle the total precipitation collected from the existing and future heap leach pads, while the barren solution pond is designed to impound all solutions including all precipitation, for a period of one year.

There are a number of advantages of the pregnant solution pond being inside the heap leach pad including:

1. A reduction in capital cost as the cost of constructing and lining a separate pond is eliminated.

2. The pregnant solution pond is not susceptible to freezing as it is inside the heap and not exposed to the cooling of the environment when temperatures are below freezing. This should extend the period of operation for the spraying system.

The barren solution pond is 42 m x 42 m x 3 m depth. Its design solution capacity is 3348 m^3 which does not include the freeboard capacity. To meet environmental standards, a cyanide recovery/destruction system will be installed to reduce cyanide effluents to the acceptable levels. Barren solution in excess of process requirements will be treated to reduce cyanide levels to regulated levels. Any precipitates produced during the effluent treatment stage will be impounded in the sludge impound ponds, and the quantity of material produced will depend on the destruction method selected. An overflow from the barren pond to the solution impound pond is provided to handle emergency run off only. The impound pond also provides solution storage capacity for retreatment of effluent should it be necessary.

The solution from the pregnant solution pond is pumped at a rate of 22 litres per second using one pump (one installed spare) located on the wall at the heap leach dam. It is pumped to the pregnant solution tank inside the process building.

Solution used for sprinkling on the heaps is pumped from the barren solution pond using two (one operating, one stand-by) submersible solution pumps located in the pond. The solution is pumped into a 2.0 m diameter x 2.0 m high heap surge tank where cyanide solution from the reagent mixing area is added. This leaching solution is pumped to the piping system on the heaps using one of two heap leach pumps. As the height of the heap increases, it will be necessary to add a second stage pump to each system.

5.6 Area 40 - Process Plant

The process plant is the central building where the metallurgical recovery takes place. The process plant is divided into five distinct areas: Area 41 - Clarification; Area 42 - De-aeration; Area 43 - Zinc Precipitation; Area 44 - Reagent Storage; Area 45 - Laboratory. The building also contains the administrative offices and the change room facilities for the operation.

The process plant is enclosed in a 24 m x 48 m prefabricated structural steel building. The building uses the natural slope of the terrain to provide an upper and lower floor plan. The uppper floor contains the office, changeroom, laboratory and warehouse facilities, while the lower area supports the maintenance facilities and the heavier process equipment.

The sloping concrete floor on the lower level drains all solutions into a common solution sump. The upper level floor is sealed and sloped to collect spilled solutions and reagents in the sump on the bottom floor. Reagents and process equipment are brought into the building through the loading dock area. Man doors, strategically placed around the building, also provide access to the building from outside. Access from the bottom floor to the second floor is provided by two flights of stairs. A three tonne crane services the equipment and reagent handling areas in the building.

5.6.01 Area 41 - Clarification

Pregnant solution is pumped from the heap to the pregnant solution surge tank in the clarification area of the process plant. The 3.0 m diameter by 3.0 m high carbon steel tank is equipped with an overflow launder to return pregnant solution to the pregnant solution pond. The overflow arrangement prevents cross contamination of barren and pregnant solutions in the event of a restriction in the process flow within the plant. The surge tank is a flat bottomed tank located on a concrete foundation on the bottom floor. The pregnant solution tank discharges directly into the clarifier feed pump. The feed pump is a centrifugal process pump rated at 23 litres per second and pumps solution to the pressure clarifier. The pump is located on a common pump/motor base supported by a concrete foundation. A stand-by clarifier feed pump is installed to allow continuous operation of the clarification system.

The leaf clarifier filter removes suspended solids from the pregnant solution before de-aeration takes place. The filter contains cloth filter leaves submerged in a circular carbon steel tank. Two or more filter leaves may be removed from the tank for repair or replacement without adverse effect on filtration. A three tonne bridge crane assists in the removal of the caked filters. A lay down area is used to service the filters with a catch basin and drain to the barren/reagent mix tank.

A system to precoat the clarifier is provided as part of the process plant.

5.6.02 Area 42 - De-aeration

Solution from the clarifier discharges into a sump from where it is pumped to the clarified solution tank on the upper floor. The clarified solution is drawn by a vacuum into the de-aeration tower. The tower is located inside the building and is fed from the clarified solution tank which provides sufficient suction head for the precipitate feed pump. Air and water vapour are collected from the de-aeration tower by a moisture trap located above the de-aeration tower. A 200 CFM vacuum pump rated @ -711 mm Hg vacuum provides the necessary vacuum for the de-aeration tower. The moisture from this trap is collected in the barometric seal tank with solution being overflowed to the ground floor sump.

5.6.03 Area 43 - Zinc Precipitation

The de-aerated pregnant solution is drawn from the de-aeration tower to the precipitate feed pump. A precisely controlled zinc dust feeder distributes the zinc dust into the zinc cone to be mixed with barren solution from the barren recycle pump. The zinc dust slurry is introduced into the de-aerated pregnant solution prior to entering the precipitate feed pump. The zinc/gold precipitate is collected on one of three plate and frame filter press located in a security area. A spare filter press is installed . The filter press is serviced by the 3 tonne electric crane. Solution flow has to be switched to allow the removal of the gold precipitate from the press to be sent to the refinery on a regular schedule.

Barren solution from the filter press flows by gravity to the barren solution tank. Two recycle solution pumps (one operating, one spare) are used to recycle the solution to the process; any excess solution to that requfired in the process is sent to the barren solution pond.

5.6.04 Area 45 - Reagent Mixing

Cyanide is received and stored in one tonne containers. It is transferred to the reagent mixing area, where the container is unloaded into the cyanide mix tank using the cyanide feeder. The mixer ensures that the cyanide is completely dissolved before it is pumped to the process addition point.

Lime is delivered to the site in bulk tanker trucks and is pneumatically conveyed to the storage bins. Two bins are used on the site. One bin is installed at the crushing plant, where a belt feeder is used to meter the lime on to the crushed ore conveyor belt. This ensures intimate mixing of lime with the ore during the ore stockpiling procedure. The second bin is located in the process building. A feeder is used to meter the line to the mix tank where an agitator ensures a homogenous mixing of the lime slurry.

5.6.05 Area 44 - Reagent Storage

Reagents are stored on the upper floor of the process building. Drums of zinc dust, diatomaceous earth bags and one tonne cyanide boxes are stored inside. All additional cyanide and lime storage is outside the process building

5.6.06 Area 45 - Laboratory

The laboratory is equipped with an atomic absorption sphectrophotometer for solution assays for gold and silver. A fire assay facility has also been provided to allow assays on solid materials. Other determinations may be completed including total cyanide concentration. The laboratory is also supplied with running water, sink, fume hood, fan, hot plate, reagents and glassware.

5.7 Area 50 - Refinery

The zinc/gold precipitate is delivered to the refinery for flux preparation before being charged into the preheated bullion furnace. The furnace is fired by an oil burner using No. 2 fuel oil. Exhaust from the bullion furnace is collected in a hood and discharged to atmosphere by an exhaust fan rated at 6.61 m^3/sec. The fan is supplied with a 40 hp motor.

The refinery building is in a secure area of the process building. It is complete with a lunch room, washroom and vault for storage of precipitate and bullion.

5.8 Area 60 - Services

The process building contains the administrative offices for the site and change facilities for the staff, as well as the potable and process water supply and the diesel generator unit for providing power to the site. The process, drinking and fire protection water supply is pumped from Seymour Creek to a storage tank located above the process plant area. Water flows from the tank by gravity to be pressurized for the particular requirement.

6.0 CAPITAL COST ESTIMATE

6.1 Introduction

A capital cost estimate is presented based on geological data, the laboratory testwork completed to date, standard North American engineering practice and the process flowsheets and plant layout drawings in the Appendices of this report. The estimate is based on the 743,200 tonne per year mining rate. The process equipment is designed to handle production for a 743,200 tonne per year heap leach operation.

For the purposes of this estimate, costs are expressed in Canadian funds, at fourth quarter 1986 pricing levels.

The capital cost is based on single or double source quotations and construction costs provided by a single contractor. The costs have been sub-divided into areas as described in the Plant Description (Section 5.0) of this report.

The capital equipment costs include fabrication, supply and delivery of new and used equipment. Provincial Sales Tax has not been included. All costs in summary have been rounded to the nearest $100.

For the purpose of the feasibility study, a capital cost estimate has been prepared which includes a ten percent contingency.

6.2 Capital Equipment and Installation Cost Estimate

The capital and installation cost estimate has been broken down by individual area and activity. In order to reduce capital costs, it has been assumed that the mining and crushing will be done by a contractor.

Area 10 - General

Equipment		
Building	$ 5,000	
Site Preparation	93,000	
Septic System	10,000	
Labour	10,000	
Area Total	$118,000	$118,000

Area 15 - Mining

By contract

Area 20 - Crushing

Equipment (lime addition)	$ 40,900	
Electrical	11,100	
Labour	11,100	
Area Total	$ 63,100	$ 63,100

Area 30 - Leaching

Site Preparation	$ 86,000	
Pad and Pond Liners (inc. Labour)	200,000	
Equipment	25,700	
Piping	8,300	
Electrical	8,300	
Pump House Building	5,000	
Foundation	1,500	
Labour	8,300	
Total	$343,100	$343,100

Area 40 - Process Plant

Site Preparation	$ 50,000	
Building	130,000	
Equipment	$411,300	
Piping	111,000	
Electrical	111,000	
Civil work	77,000	
Labour	108,000	
Total	$998,300	$998,300

Area 50 - Refinery/Laboratory

Equipment	$156,500	
Piping/Ductwork	42,200	
Electrical	42,200	
Labour	42,200	
Total	$283,100	$283,100

Area 60 - Services

Equipment	$225,000	
Piping/ductwork	58,000	
Electrical	58,000	
Labour	58,000	
Total	$399,000	$399,000

Capital Cost Summary

Total estimated equipment supply and construction	$2,204,600
Contingency @ 10%	$ 225,400
Sub total	$2,430,000
Engineering and construction supervision	$ 220,000
TOTAL ESTIMATED CAPITAL COSTS	$2,650,000

7.0 OPERATING COST ESTIMATE

7.1 Introduction

The operating costs have been calculated for the commercial operation of the heap leaching joperation on a cost per tonne and cost per year basis. Reagent consumptions have been calculated from the overall plant mass balance as presented on the flowsheets. All costs have been rounded up to the nearest dollar and are on a 30 day per month basis. All costs include freight where applicable.

7.2 Mining

This cost is a quotation from the mining contractor.

	Ore mining	$1.25 per tonne
	Ore haulage	$2.45 per tonne
7.2.01	Total ore mining	$3.70 per tonne
	Waste mining	$1.25 per tonne
	Waste haulage	$1.28 per tonne
7.2.02	Total Waste Mining	$2.53 per tonne

7.3 Crushing

As per contractors quotation.

7.3.01	Crushing cost, 743,200 tonnes @ $1.85 per tonne	$1,374,900

7.4 Leaching Cost

7.4.1 Reagents and Consumables

	Reagent/ Consumable	Unit Cost	Usage	Cost per tonne
7.4.01	Sodium Cyanide	$1.95/kg	2.0 kg/tonne	$3.90
7.4.02	Lime	0.382/kg	4 kg/tonne	1.52
7.4.03	Zinc Dust	3.10/kg	7 g/tonne	0.03
7.4.04	Diatomaceous Earth	0.81/kg	25 g/tonne	0.02
7.4.05	Filter Cloth	1.50/m	0.0036 m /tonne	0.01
7.4.06	Laboratory Reagents	allow		0.02
7.4.07	Refinery Reagents	allow		0.06
7.4.08	Total Reagents and Consumables			$5.56

7.4.2 Supplies and Maintenance Materials

Allow 10% of capital costs	$80,500
Cost per tonne of ore	$0.11

7.4.3 Power Costs

400 kW @ $0.20 per kWh	$80
180 days per year, 24 hr per day, cost	$345,600
50 kW @ $0.20 per kWh	
180 days per year, 24 hr per day, cost	$43,200
Power cost per tonne	$0.52
Annual Cost	$388,800

7.4.4 Crushed Ore Haulage

As per contractor's cost estimate

Cost of crusher ore haulage 743,200 tonnes @ $0.55 per tonne	$408,800

7.4.5 Stack Ore on Leach Pad

As per contractor's cost estimate

Cost of stacking ore in feed $743,200 tonne @ $0.10 per tonne	$ 73,300

7.4.6 Staffing Requirements

Staffing for the project would consist of 8 permanent and 11 seasonal employees as below.

1 General Superintendent	@ $55,000 per year	$ 55,000
1 General Foreman	@ $40,000 per year	40,000
8 Operators, 2 per shift, 4 shifts per week, for 26 weeks leaching	@ $15/hour	199,700
1 Metallurgist	@ $3,500/month	42,000
1 Technician/Assayer	@ $3,000/month	36,000
2 Heap Operators 8 hr/day, 5 days/week	@ $14/hour	29,100
1 Diesel Mechanic	@ $20/hour, 5 days/week	41,600
2 Mechanics	@ $18/hour, 5 days/week	76,800
1 Electrician	@ $20/hour, 5 days/week 26 weeks/year	20,800
1 Clerk	@ $30,000	30,000
Labour Cost		$571,000
Fringe Benefits @ 25%		143,200
Total Labour Cost		$714,000
Tonnes leached		743,200
Labour cost per tonne		$0.96

7.4.7 Total Leach Cost

7.4.1	Reagents and Consumables	$5.56	$4,132,200
7.4.2	Supplies and Maintenance Materials	0.11	80,500
7.4.3	Power	0.52	388,800
7.4.4	Crushed Ore Haulage	0.55	408,800
7.4.5	Ore Stacking	0.10	73,300
7.4.6	Staffing	0.96	714,000
7.4.7	Total Leach Cost	$7.80	$5,797,600

7.5 Administration Costs

7.6.01	Heating	$ 30,000
7.6.02	Lighting	5,000
7.6.03	Truck, Stationwagon Lease	8,000
7.6.04	Office Supplies	12,000
7.6.05	Telephone	30,000
7.6.06	Outside Services	10,000
7.6.07	Safety Supplies	12,000
7.6.08	Miscellaneous Supplies 20% of above	21,800
	Total Administration	$ 128,800
	Tonnes of ore treated	743,200
	Cost per tonne of ore	$ 0.18

7.6 Overhead Costs

For head office overhead distribution allow $0.10 per tonne	$ 75,000

7.7 Summary of Operating Costs

		Cost per Tonne of Ore Hauled	$
7.7.01	Mining ore	$ 3.70	2,749,800
7.7.02	Mining waste	0.71	526,500
7.7.03	Crushing	1.85	1,374,900
7.7.04	Leaching cost	7.80	5,797,600
7.7.05	Administration cost	0.18	128,800
7.7.06	Overhead	0.10	75,000
		$14.34	10,652,600

8.0 SCHEDULE

The schedule for the installation of the Antoniuk heap leach requires 42 weeks for approval of environmental and mining permits, design, procurement, installation, mining, crushing and commissioning. A very closely controlled and achievable schedule is required in order to commence heap leaching in April. The heap leach and pond liners must be installed during relatively warm weather in order to properly deploy and weld the HDPE at minimum cost. The process building should also be complete in order to facilitate the installation of process equipment during the winter and the commissioning and operation of the heap leach in early April. One complete 743,200 tonne leach may be completed before winter shut down in October. The proposed duration of each critical activity is presented in Table 5. The proposed schedule is shown in Figure 5.

Table 5

DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY

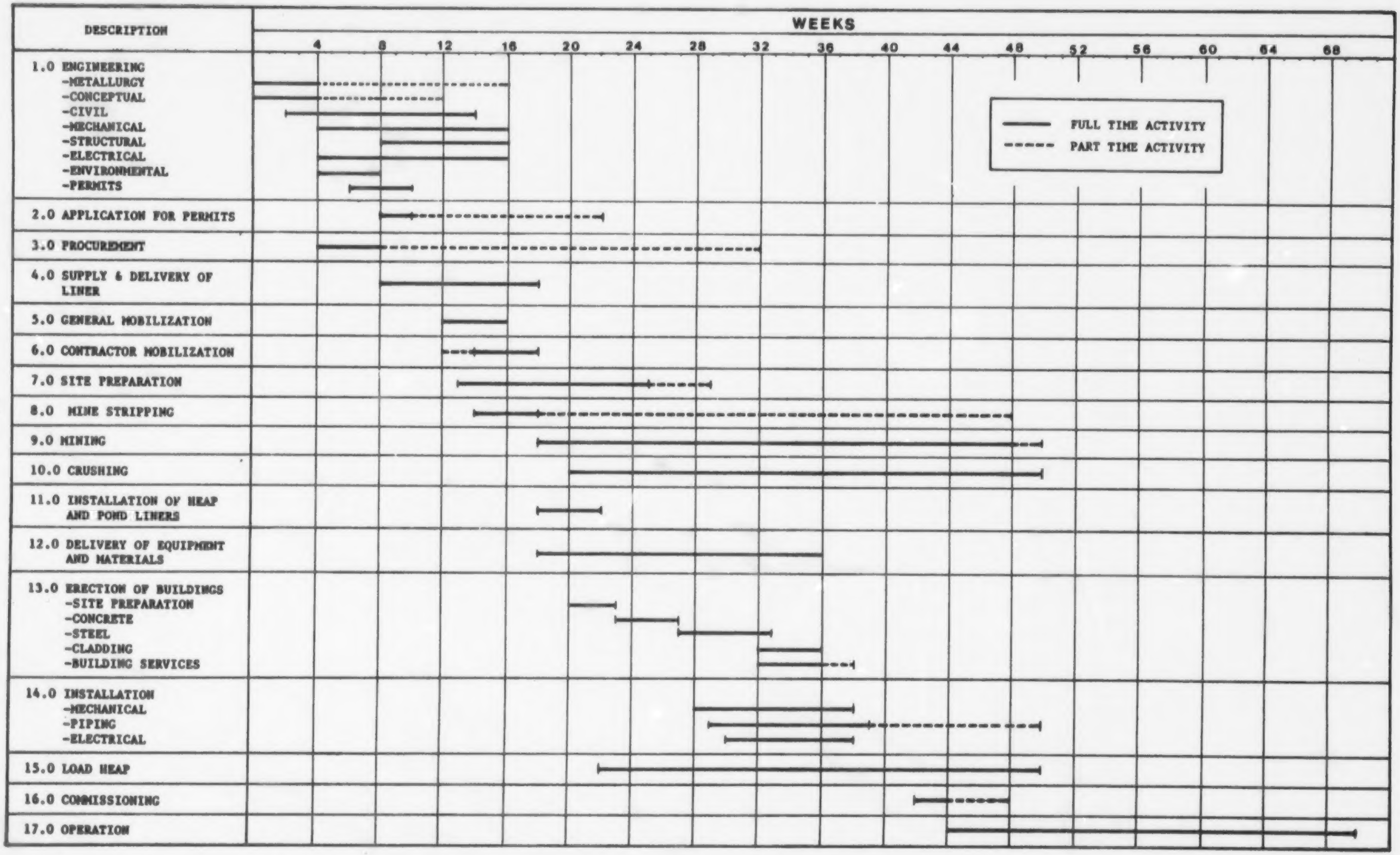
ENGINEERING, CONSTRUCTION AND COMMISSIONING PROJECT SCHEDULE

Activity		Full Time (Weeks)	Duration Part Time (Weeks)	Total (Weeks)
1.0	Engineering			16
	- Metallurgy	4	12	16
	- Conceptual	4	8	12
	- Civil	12	0	12
	- Mechanical	12	0	12
	- Structural	8	0	8
	- Electrical	12	0	12
	- Environmental	4	0	4
	- Applications	4	0	4
2.0	Submission of Applications (Environmental, Mining, etc. Permits)	2	12	14
3.0	Procurement	4	24	28
4.0	Supply and Delivery of Liner Material	10	0	10
5.0	General Site Mobilization	4	0	4
6.0	Contractor Mobilization	4	0	4
7.0	Site Preparation	12	4	16
8.0	Mine Stripping	4	30	34
9.0	Mining	30	2	32
10.0	Crushing	30	0	30
11.0	Installation of Heap and Pond Liners	4	0	4
12..0	Delivery of Equipment and Construction Materials	18	0	18
13.0	Erection of Buildings			18
	- Site Preparation	3	0	3
	- Concrete	4	0	4
	- Steel	6	0	6
	- Cladding	4	0	4
	- Building Services	4	2	6
14.0	Installation			
	- Mechanical	8	0	8
	- Piping	10	11	21
	- Electrical	8	0	8
15.0	Load Heap	28	0	28
16.0	Commissioning	2	4	6
17.0	Operation			26

Figure 5

DAWSON RANGE HEAP LEACH PROJECT
YUKON TERRITORY

ENGINEERING, CONSTRUCTION AND COMMISSIONING PROJECT SCHEDULE



9.0 PRELIMINARY FINANCIAL ANALYSIS

A preliminary financial model for the cash flow anticipated from the Antoniuk property was prepared on a computer spreadsheet program. The information for the design criteria has been developed as described earlier in this report and is summarized in Table 6. Using these criteria, the cash flow statement was prepared for this base case and is presented in Table 7. Sensitivity of the major criteria of gold price, head grade, gold heap leach recovery, and operating cost are presented in Table 8.

Using the base design criteria, the project does not create sufficient cash flow from the gold recovered to pay for the operating costs. The major factors contributing to this are:

1. The gold price would have to be in excess of US$500 per ounce in order to yield a positive internal rate of return for the development of the property. The recent increase in gold prices to the US$450 range provides encouragement for the improvement in economics from this parameter.

2. Gold recovery would have to increase by 10% from the basic design criteria value to a level of 85% for the operation to cover its operating costs. However, improved recovery would have to be in excess of 85% to provide a return on the invested capital. Pilot plant column leach tests did indicate the feasibility of high recoveries, but the design criteria were designated at conservative levels. Should further testwork verify the higher gold extractions, then a higher gold recovery figure could be used for the design criteria.

3. The low grade of the ore. A 0.99 gram per tonne feed grade is equivalent to 0.029 oz of gold per short ton. This head is low even by south-western U.S.A. standards. The gross value of gold in the feed, assuming US$400 per ounce, is $17.05 per tonne. In order to break even on the operating cost of $14.34 per tonne, recovery of gold in the process would have to be in the order of 85%.

   The grade of the deposit would have to be 1.5 grams per tonne in the first 3.5 years of operation to achieve an internal rate of return of 25%.

4. Operating costs for the heap leaching are high. The topography of the area requires that the plant be located a distance from the mine. This increased haulage distance incurs a cost of $1.27 per tonne in excess of the "normal" haulage cost for the pit waste.

   This ore consumes 2.0 kg of sodium cyanide per tonne of ore treated. The cost of cyanide is 27.5% of the total operating cost. Heap leach operations typically consume 0.5 kg to 1.0 kg per tonne of cyanide per tonne of ore treated. Laboratory investigations to date have not been successful in reducing the consumption of cyanide to less than 2.0 kg per tonne of ore. The Antoniuk ore also requires a higher lime addition than usual to achieve the lower cyanide consumptions used in the preparation of the operating cost.

   A review of the mining of both ore and waste should be undertaken to determine whether further savings can be made in the mining program.

5. The ore reserves do not permit economics of scale of the operation that reduce operating costs to a level where profitable operations can be achieved. Increasing the plant throughput would reduce the unit cost for such items as labour, mining and haulage and other fixed costs of the operation. An exploration drilling program to delineate increased ore reserves should be considered.

6. The internal rate of return for the project is not sensitive to the capital cost under the base design criteria specified. In order to minimize capital costs for the project, the crushing of the ore for heap leaching the transportation of the ore and subsequent stockpiling will be undertaken by the mining contractor. Further reductions in capital cost are feasible by the purchase of refurbished equipment rather than new equipment. As a general rule, the cost of refurbished equipment is approximately 50% that of new equipment. The cost for installation, piping, electrical and similar services does not change from the base case cost. Savings in the range of $350,000 would be possible in the project by the purchasing of refurbished equipment.

10.0 RECOMMENDATIONS FOR FUTURE WORK

Although the base case initial design criteria indicates that the development of the Antoniuk property is not economically viable at the present time, there are a number of opportunities for improving the economics of the project. These include the verification of higher than presently designated gold recovery, reductions in cyanide consumption, and a review of the mining costs for the project with the objective of achieving the lowest possible cost. The investigation of increased ore reserves in the area, so that advantage can be taken of the economics of scale of larger operations, is of particular importance to achieve economic development of the property.

The metallurgical and mineralogical investigations of samples from the deposit failed to determine specific causes for high lime and cyanide consumptions on the mineral associations tested. Further testing using increased lime additions to reduce cyanide consumptions could result in the improved economics for the property. The testing of selected samples from the various mineralogical and geological associations from the ore body should also be undertaken to ensure that the sample tested is representative of the Antoniuk deposit.

Table 6

ARCHER CATHRO - ANTONIUK PROPERTY

PRO FORMA FINANCIAL STATEMENTS

BASE CASE DESIGN CRITERIA USED FOR FORECAST PREPARATION
Costs in Canadian dollars unless stated otherwise

Ore reserves	3716000 tonnes
Daily mining rate	3097 tonnes per day
Operating days	240 per year
Waste to ore ratio-years 1-3.5	0.28 tonnes of waste per ton of ore
-years 3.5-5	2.2 tonnes of waste per ton of ore
Mine life	5 years
Ore grade-years 1-3.5	0.990 grams per tonne
-years 3.5-5	1.5 grams per tonne
Recovery	75 percent
Gold price	$400 U.S. per oz
Exchange rate	$0.72 Canadian per $US
Annual inflation rate	3.5%
Ore mining and haulage cost	$2.70 per tonne of ore
Waste mining and haulage cost	2.50 per tonne of waste
Crushing cost	1.85 per tonne of ore
Leaching cost	$7.90 per tonne of ore
Refining cost	$3.50 per oz of gold
Administration Cost	$0.18 per tonne of ore
Overhead	$0.10 per tonne of ore
Exploration	As shown
Depreciation rate	30.0% declining balance
Mining tax rate	6.0% of operating income less exploration expense
Corporate tax rate	30.0% of operating income less deductions
Capital investment	2.65 million

PROFIT AND LOSS STATEMENT

Table 7

	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	TOTALS
Ore mined-tonnes	371600	743200	743200	743200	743200	371600	3716000
Waste mined-tonnes	104048	208096	208096	208096	1635040	817520	3180896
Ore leached	185800	743200	743200	743200	743200	557400	3716000
Ore grade-grams per tonne	0.99	0.99	0.99	0.99	1.5	1.5	
Recovery-%	75	75	75	75	60	60	
Gold produced-grams	137957	551826	551826	551826	668880	501660	2963975
Total Gold produced-oz	4430	17721	17721	17721	21480	16110	95182
Gold price per oz-$US	400	400	400	400	400	400	
Exchange rate	0.72	0.72	0.72	0.72	0.72	0.72	
Gold sales	2461224	9844894	9844894	9844894	11933205	8949904	52879014
Operating costs							
Mining	1374920	2846084	2945697	3048797	3155505	1632974	15003977
Waste	263241	544910	563982	583721	4746902	2456522	9159278
Crushing	687460	1423042	1472849	1524398	1577752	816487	7501988
Milling	2898480	5999854	6209848	6427193	6652145	3442485	31630005
Refining	15506	64194	66440	68766	86270	66967	368142
Total operating costs	5239607	10878084	11258817	11652875	16218574	8415434	53663391
Operating profit	-2778384	-1033190	-1413923	-1807981	-4285369	534469	-10784377
Expenses							
Administration	66888	138458	143304	148320	153511	79442	729923
Overhead	37160	76921	79613	82400	85284	44134	405513
Exploration		0	100	100	100	100	400
Depreciation	795000	556500	389550	272685	190880	133616	2338230
Total expenses	899048	771879	612568	503505	429774	257292	3474066
Operating income	-3677432	-1805069	-2026490	-2311486	-4715144	277177	-14258443
Cumulative operating income		-5482501	-7508991	-9820477	-14535620	-14258443	
Taxes							
Mining tax	0	0	0	0	0	0	0
Corporate tax	0	0	0	0	0	0	0
Total taxes payable	0	0	0	0	0	0	0
Net Income	-3677432	-1805069	-2026490	-2311486	-4715144	277177	-14258443
Cumulative net income	-3677432	-5482501	-7508991	-9920477	-14535620	-14258443	
Add depreciation	795000	556500	389550	272685	190879.5	133615.65	2339230
Available funds	-2882432	-1248569	-1636940	-2038801	-4524264	410793	-11920213
Cumulative funds	-2882432	-4131001	-5767941	-7806742	-12331006	-11920213	
Capital expenditure	2650000						
Cash flow	-5532432	-1248569	-1636940	-2038801	-4524264	410793	
Net present value	-10068675						
Internal rate of return	-91%						

Table 8

```
                          SENSITIVITY ANALYSIS

                     ------------------------------
                       GOLD    INTERNAL     NET
                      PRICE      RATE     PRESENT
                       $US      RETURN     VALUE
                     ==============================


                        300    -128.0% -17940820
                        350    -112.5% -14004748
 PRESENT      →         400     -91.3% -10068675
  PRICE                 450     -57.7%  -6132603
                        500      -9.2%  -2196531
                        600      53.0%   3306601
                     ******************************



                     ------------------------------
                       HEAD    INTERNAL     NET
                      GRADE      RATE     PRESENT
                     GRAM/TONNE RETURN     VALUE
                     ==============================


                        0.8     -91.6% -14202874
 PRESENT                0.9     -91.4% -12026980
  HEAD        →           1     -91.3%  -9851086
                        1.1     -91.1%  -7675192
                        1.2     -90.9%  -5499298
                        1.3     -90.7%  -3265510
                        1.4     -90.6%  -1818655
                        1.5      24.6%    333571
                        1.6      56.9%   1736230
                     ******************************



                     ------------------------------
                       GOLD    INTERNAL     NET
                     RECOVERY    RATE     PRESENT
                        %       RETURN     VALUE
                     ==============================

                         70    -106.2% -12315628
 PRESENT      →          75     -91.3% -10068675
 RECOVERY                80     -71.6%  -7821722
                         85     -45.8%  -5574769
                     ******************************



                     ------------------------------
                     OPERATING INTERNAL     NET
                       COST      RATE     PRESENT
                        $       RETURN     VALUE
                     ******************************

                      10.00      19.6%    294226
 PRESENT              12.00     -48.2%  -4185499
  COST        →       14.34     -91.3% -10068675
                      16.00    -105.6% -14242210
                      18.00    -117.5% -19270566
```

APPENDIX I

Equipment Ledger

ANTENIUK HEAP LEACH PROJECT

EQUIPMENT LEDGER

EQUIPMENT NO	NO	DESCRIPTION	CAPACITY\ SIZE	UNIT COST	TOTAL COST	RUNNING HP
AREA 20 CRUSHING						
20-GR-001	1	Grizzly	200tph	10000	10000	
20-VF-001	1	Feeder Plant	200tph	105000	105000	
20-JC-001	1	Jaw Crusher & discharge conveyor	Rexnord 2540	215000	215000	
20-RB-001	1	Hydraulic Rock Breaker	1000ftlbs	51200	51200	
20-VF-002	1	Surge bin/cone crusher feeder	20 cu yd	75000	75000	
20-CC-001	1	Symons Cone Crusher	4.25 ft	250000	250000	
20-TB-002	1	Truck loading bin	20 cu yd	75000	75000	
20-MG-001	1	Diesel generator set	500 kW	22000	22000	
20-TK-001	1	Lime Storage Bin		20000	20000	
20-SF-001	1	Lime Screw Feeder with Bin Activator		13900	13900	
20-DC-001	1	Dust Control Unit		7000	7000	
AREA 20 CRUSHING TOTAL				844100	844100	
AREA 30 LEACHING						
30-PP-001/2	2	Pregnant Solution Feed Pump	Flygt 2102	3100	6200	7.5
30-PP-003/4	2	Heap Leach Pump	3x2x10 Worthington 1750 RPM	5100	10200	30
30-PP-005/6	2	Barren solution bleed pump	Flygt 2102	3100	6200	7.5
30-PP-007	1	Effluent feed pump	Flygt 2102	3100	3100	7.5
AREA 30 LEACHING TOTAL				14400	25700	52.5
AREA 41 CLARIFICATION						
41-TK-003	1	Clarified Preg. Holding Tank	3mx3m	6600	6600	
41-PP-001/2	2	Clarifier Feed Pump	4x3x13Worthington 1750 rpm	2300	4600	30
41-TK-004	1	Precoat Mix Tank	1.5mx1.5m	1500	1500	
41-MX-002	1	Agitator	Lightnin'	1400	1400	0.3

ANTONIUK HEAP LEACH PROJECT

EQUIPMENT LEDGER

EQUIPMENT NO	NO	DESCRIPTION	CAPACITY/ SIZE	UNIT COST	TOTAL COST	RUNNING HP
41-PP-007	1	Precoat Pump	4x3x13Worthington 1750 rpm	2300	2300	5
41-TK-001	1	Pregnant Solution Holding Tank	3mx3m	6600	6600	
41-PP-005A/B	2	Clarified Preg. Pump	4x3x13Worthington 1750 rpm	2200	4400	7.5
41-CL-001/2	2	Pressure Clarifier		100000	200000	
AREA 41 CLARIFICATION TOTAL				122900	227400	42.5
AREA 42 DEAERATION						
42-TW-001	1	Deaeration Tower	1.5mx3.5m	16700	16700	
42-TK-002	1	Moisture Trap		2200	2200	
42-PP-001	1	Vacuum Pump	200cfm	5200	5200	20
AREA 42 DEAERATION TOTAL				24100	24100	20
AREA 43 ZINC PRECIPITATION						
43-FR-001	1	Zinc Dust Feed System		6500	6500	0.3
43-PP-001	2	Filter Press Feed Pump	4x3x13 Worthington 1750 RPM	5040	10080	30
43-FP-001	3	Filter Press		30000	90000	
43-TK-002	1	Barren Solution Tank	3mx3m	6600	6600	
43-PP-002	2	Barren Solution Pump	4x3x13 Worthington 1750 RPM	5040	10080	7.5
43-SP-001	1	Sump Pump	1.5" vertical	2300	2300	2
AREA 43 ZINC PRECIPITATION TOTAL				55480	125560	39.8
AREA 45 REAGENT MIXING						
45-TK-001	1	Cyanide Holding Tank	2.5mx2.5m	5000	5000	
45-MX-001	1	Cyanide Tank Agitator		3400	3400	2
45-FR-001	1	Cyanide Feeder		2000	2000	0.3
45-TK-002	1	Lime Preparation Tank		16000	16000	
45-MX-002	1	Lime Tank Agitator		inc		2

ANTONIUK HEAP LEACH PROJECT

EQUIPMENT LEDGER

EQUIPMENT NO	NO	DESCRIPTION	CAPACITY/ SIZE	UNIT COST	TOTAL COST	RUNNING HP
45-TK-003	1	Lime Bin		170		
	2	Exhaust Fan		1500	3000	1
45-PP-007	2	Cyanide Solution Feed Pump		1200	2400	[illegible]
45-PP-011	2	Lime Slurry Feed Pump		1200	2400	[illegible]
AREA 45 REAGENT MIXING TOTAL				[illegible]	[illegible]	[illegible]
AREA 51 REFINERY LABORATORY						
	1	Lot Laboratory Equipment		114000	114000	
	1	Atomic Absorption Unit		15000	15000	
51-BF-001	2	Bullion Furnaces		15000	30000	7
51-EF-001	1	Exhaust Fan		750	750	45
AREA 51 REFINERY LABORATORY TOTAL				[illegible]	[illegible]	52
AREA 60 SERVICES						
	1	Office equipment		10000	10000	
60-PP-001	1	Potable Water Pump		4000	4000	10
60-PP-002/3	2	Process Water Pump		4000	8000	10
60-PP-003	1	Diesel Power Fire Water Pump		7900	7900	
60-PP-00[illegible]/3	2	Seymour Creek Pump		7700	15400	125
60-TK-001	1	Raw Water Storage Tank	5mx5m	12000	12000	
60-BO-001	1	Hot Water Boiler		2300	2300	
60-CR-001	1	Air Compressor		10000	10000	15
60-DR-001	2	Instrument Air Drier		1400	2800	
60-TK-004	1	Diesel Fuel Tank		6600	6600	
60-MG-001	1	Diesel Generator Set	500kW	136000	136000	
		Office equipment		10000	10000	

```
                                   ANTONIUK HEAP LEACH PROJECT

                                   EQUIPMENT LEDGER

----------------------------------------------------------------------------------------
EQUIPMENT    NO             DESCRIPTION            CAPACITY:           UNIT   TOTAL  RUNNING
   NO                                                      SIZE        COST   COST      HP
========================================================================================

AREA 60 SERVICES TOTAL                                                211900  225000     160
****************************************************************************************


PROCESS EQUIPMENT TOTAL                                                       818460   778.4

****************************************************************************************
```

APPENDIX II

Quotations

5209

[illegible]
[illegible]

[illegible]
[illegible]
[illegible] LEACHING PROJECT - ANTONINA

1.	MINING	1,374,900 MTPY	1.25	1,718,625
2.	ORE HAULAGE	743,200 MTPY	2.45	1,820,840
3.	WASTE HAULAGE	631,700 MTPY	1.28	808,576
4.	CRUSHED ORE HAULAGE	743,200 MTPY	0.55	408,760
5.	STACK ORE ON LEACH HEAP	743,200 MTPY	0.10	74,320
6.	EXCAVATE AND MOVE ROCK	64,340	5.00	321,700
7.	EXCAVATE AND MOVE EARTH	9,100	2.50	22,750
8.	ROAD CONSTRUCTION	1.86 KM	50,000	93,000
9.	SUPPLY PLACE AND COMPACT IMPERVIOUS SAND UNDER (A) LEACH PAD LINERS (B) BARREN POND (C) [illegible] POND (D) COLLECTION DITCHES	41,600 M3	12.00	479,200
10.	SUPPLY AND PLACE COARSE SAND ABOVE GEOTEXTILE	24,000 M	9.00	216,000
11.	SUPPLY AND PLACE FINE GRAVEL ABOVE LEACH PAD LINERS	16,000 M3	9.00	144,000
12.	SUPPLY AND PLACE COARSE GRAVEL ABOVE LEACH PAD LINERS	16,000 M3	8.00	128,000
13.	SUPPLY AND PLACE DAM FILL	23,000 MT	2.00	46,000
14.	SUPPLY AND PLACE COARSE GRAVEL AROUND			
	(A) LEACH PAD LINER	540 M3	10.00	5,400
	(B) BARREN POND XXX POND LINER	150 M3	10.00	1,500
	(C) [illegible] POND LINER	50 M3	10.00	500
TOTAL ESTIMATED COST				6,307,171

TAXES INCLUDED IN THE
XXX THE ABOVE

(A) FEDERAL	0
(B) PROVINCIAL	0
COST TO FURNISH PERFORMANCE BOND OF 2,000,000	20,000

1 IN ADDITION WE WOULD BE PREPARED TO CRUSH THE ORE FOR 1.85 PER TONNE PLUS 30,000 MOB.

2 ABOVE DOES NOT INCLUDE ROOM AND BOARD. WE HAVE FULLY EQUIPPED CAMP, OFFICES, ETC AVAILABLE.

[illegible]
[illegible]

[illegible]

[illegible]

TELEX RECV CONNECTED 16-Oct-86 14:51 04

WITTEDEV MSGA

NORDBERG GLPH

OCT 16 86 2:30 PM

ATTN: MIKE YETMAN

RE: WHITTECK PROJECT 5209

FOLLOWING IS OUR ESTIMATE OF EQUIPMENT REQUIRED FOR THE ABOVE PROJECT. TO CRUSH 160 TONS PER HOUR 22'' TOP FEED SIZE DOWN TO MINUS 1-1/2'':

1. PORTABLE FEEDER PLANT
   - CHASSIS MOUNTED 15 CUBIC YARD FEED HOPPER WITH VARIABLE SPEED DISCHARGE BELT CONVEYOR TO FEED JAW CRUSHER
   105,000.

2. PORTABLE JAW PLANT
   - CHASSIS MOUNTED 2540 NORDBERG JAW CRUSHER WITH DISCHARGE CONVEYOR TO FEED SURGE BIN
   215,000.

3. PORTABLE SURGE BIN/CONE CRUSHER FEEDER
   - CHASSIS MOUNTED 20 CUBIC YARD SURGE HOPPER WITH DISCHARGE CONVEYOR TO FEED ~~CONE~~ CONE CRUSHER
   75,000.

4. PORTABLE CONE PLANT
   - CHASSIS MOUNTED 4-1/4 FT. STANDARD SYMONS CONE CRUSHER WITH DISCHARGE CONVEYOR
   250,000.

5. PORTABLE TRUCK LOADING PLANT
   - CHASSIS MOUNTED 20 CUBIC YARD HOPPER WITH DISCHARGE CONVEYOR FOR LOADING TRUCK
   75,000.

5.A OPTIONAL TRUCK LOADING BIN
   - 150 FT. LONG STATIONARY CONVEYOR FEEDING 150 CUBIC YARD BIN FOR TRUCK LOADING (FOUNDATIONS NOT INCLUDED)
   120,000.

5.B OPTIONAL TRUCK LOADING RECLAIMER
   - STORAGE PILE RECLAIMING TUNNEL WITH TRUCK LOADING CONVEYOR
   130,000.

6. POWER VAN (USED VAN) WITH CONTROL CABIN, DIESEL GENERATOR SET (TO SUPPLY 500 KW), MOTOR STARTERS, AND WIRING

220,000.

ALL PRICES IN CANADIAN DOLLARS F.O.B. SASKATOON, TAXES NOT INCLUDED.

PLEASE CALL ME IF YOU NEED FURTHER INFORMATION FOR THE ABOVE PROJECT.

LARRY ERLENDSON

CC: P.KLEIN
NORDBERG GLPH

WITTEDEV MSGA

WITTEDEV MSGA

DISCONNECTED 18 OCT 88 14:56 01 MSG 8

SACAN BELV
9:23 AM

0

SEPT 25/86

ATTN: MIKE YETMAN

WE ARE PLEASED TO QUOTE BUDGET PRICES FOR BELT CONVEYORS AS FOLLOWS:

NO. 001: 24'' X 20 FT. HORIZONTAL UBPPMVUGDLY.PWC // 33 FT. HORIZONTAL 19,000.00 DLS 35.30 ft

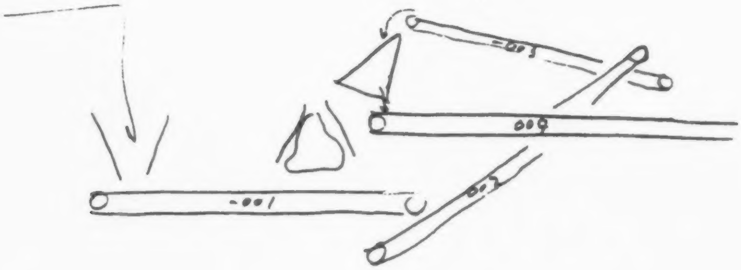
NO. 003: 24'' X 39 FT. X 13 FT. LIFT (TRUSS + WALKWAY CONSTRUCTION)
24,000.00 DLS 6'5

NO. 004: 18'' X 246 FT. HORIZONTAL (STRINGERS + LEGS CONSTRUCTION)
C/W INTERMEDIATE PLOW AND STD. DISCHARGE CHUTE AT
HD. END 78,000.00 DLS [illegible]

FOB BELLEVILLE, TAXES EXTRA

REGARDS

H MAYER
STEPHENS-ADAMSON BELLEVILLE
06-62229
SACAN BELV



TELEX RECV CONNECTED 05-Sep-86 09:27 58

WITTE DEV MSGA

T D ROOKE TOR

SEPT. 5/86 9:25 A.M.
ATTN. M. YETMAN

RE: HEAP LEACH BUDGET PRICES

1)	DEAERATION TOWER	16,700 DLRS EACH.
2)	41 TK 002 10' X 9'	5,500 DLRS EACH.
3)	CLARIFIER INCL. BODY	~~80,000 DLRS EACH~~.
4)	42 TK 002 INCL. LEG	2,200 DLRS EACH.
5)	NACN TK 8 X 8	5,500 DLRS EACH.
6)	LIME MAKE-UP SYSTEM (1 SHIFT)	14,000 DLRS EACH.
7)	DE TANK 4 X 4	2,000 DLRS EACH.
8)	DE AGITATOR 1/3 HP	1,400 DLRS EACH.
9)	PERRIN PRESSES TWO AT	30,000 DLRS EACH.
10)	41TK002 AGITATOR 2 HP	7,000 DLRS EACH.
11)	NACN AGITATOR 2 HP	7,000 DLRS EACH.

FURTHER INFORMATION WILL BE MAILED.

BEST REGARDS,

D. A. JONES
06-218451
T D ROOKE TOR

WITTE DEV MSGA

DISCONNECTED 05-Sep-86 09:30 40 MSG 31 -

PRECISIONEERING LTD

consistently reliable reinforced plastic corrosion-proof products through . . . experience . . . quality control . . . engineering

303 NANTUCKET BOULEVARD, SCARBOROUGH, ONTARIO M1P 2P2 • TELEPHONE (416) 751-9200 · TELEX 06-963795

Please Refer To File
H639-245

September 11, 1986

Witteck Development
2640 South Sheridan Way
Mississauga, Ontario
L5J 2M4

Attention: Mr. Mike Yetman

Gentlemen:

Thank you for your enquiry. After careful consideration of your requirements, we are pleased to offer the following for your consideration:

One only 5'-0" diameter x 11'-6" straight side Deaeration tower for full vacuum design.

Budget Price: $16,700.00

Accessories:
- Dished heads
- Support legs
- (4) 2"Ø flanged nozzles
- (2) 1"Ø flanged nozzles
- (4) 2" thick demister pads
- (1) 20"Ø Manway
- (1) Distributor header
- (3) Lift lugs

One only 10'-0" diameter x 9'-8" straight side lead Nitrate Mix tank

Budget Price: $ 5,500.00

Accessories:
- Flat bottom, Open top
- (4) 2"Ø flanged nozzles
- (3) Lift lugs
- (4) Hold down lugs
- (1) Mixer bridge

Terms or conditions as stated on reverse side are part of this proposal.

FANS ☐ SCRUBBERS ☐ PIPE ☐ HOODS ☐ DUCT ☐ TANKS ☐ EXHAUST SYSTEMS

MK

M·K PROCESS EQUIPMENT LTD.

RECEIVED SEP 24 1986

September 22, 1986

M-K Ref P-574

Witteck Development Inc.
2640 South Sheridan Way
MISSISSAUGA, Ontario
L5J 2M4

ATTN: Mike Yetman

Dear Sir:

We are pleased to confirm our telexed quotation of Sept. 19/86 covering a Liquid Ring Vacuum Pump system.

This package is designed to drain all the sealing water to the sewer through the separator.

For this service 200 CFM at 28"Hg., we have selected the Model PL52/2 which will deliver 220 CFM at 28"Hg., requiring 14.7 BHP. The maximum BHP required is 16 and we are offering a 20 HP motor for this service.

We quote as follows:

ONE (1) ONLY Model PL52/2 L.R.V.P., 1750 RPM, complete with:

- Baseplate under pump, motor and separator.
- Coupling.
- Coupling guard.
- 20 HP/1800/575/3/60 TEFC motor.
- Water/Air separator.
- Piping between pump and separator.
- Vacuum gauge and control valves for sealing water.

PRICE COMPLETE	$5,496.00

Price is exclusive of sales taxes, F.O.B. M-K Mississauga, terms NET 30 days. Shipment 4 to 6 weeks.

5959 SHAWSON DRIVE, MISSISSAUGA, ONTARIO L4W3Y2 (416)678-2019

Robertson
Building Systems

RECEIVED DEC 5 1986

December 1, 1986

Mr. M. Yetman, P. Eng.
Witteck Development Inc.
2640 South Sheridan Way
MISSISSAUGA, Ontario
L5J 2M4

Dear Mr. Yetman:

As per our brief, recent telephone conversation, I am pleased to submit a budget price for the "process building" for your feasibility study in the Yukon.

The attached Building Description will identify the basic building parameters and also indicates the "extras" for suitable insulation.

In summary, the budget price for the basic Steel Building Shell,
uninsulated is..$55,800.
Extra for freight is..................................$15,000
Extra for 4" RTS wall insulation c/w interior liner
and thermal clip and 6" roll down metal building
insulation in roof is.................................$43,345.
(Including Freight)

I trust this information is helpful for your feasibility study, and we look forward to working with you when a decision is made to proceed.

Yours truly,

ROBERTSON BUILDING SYSTEMS

W. N. Evoy
General Manager

WNE/ah
(attach.)

Robertson Building Systems
A Division of H. H. Robertson Inc.
61 Burford Road
M.P.O. Box 100
Hamilton, Ontario, Canada L8N 3B6
Telephone: 416-544-2831 Telex: 061-8610

CSSBI
FABRICATOR MEMBER
CANADIAN
SHEET
BUILDING INSTITUTE

CROTHERS
CATERPILLAR
CROTHERS LIMITED, ONE CROTHERS DRIVE, P.O. BOX 5511, CONCORD, ONTARIO L4K 1E2 (416) 667-5511

QUOTATION

8609-6239

Witteck Development Inc.
2640 South Sheridan Way
Mississauga, Ontario
L5J 2M4

Attention: Mr. M. Yetman

Subject: Budgetary Pricing
Trailerized Caterpillar
Generator Sets

DATE October 8, 1986
SALES TAX (FEDERAL) Extra
SALES TAX (PROVINCIAL) Extra
TERMS As specified
F.O.B. POINT Crothers, Concord
DELIVERY 22-24 weeks

CROTHERS LIMITED

BY Luigi Colangelo

ITEMS QUOTED ARE SUBJECT TO PRICES IN EFFECT AT TIME OF SHIPMENT.

QUANTITY	DESCRIPTION	UNIT PRICE	EXTENSION
	We are pleased to have the opportunity to quote the following Caterpillar generator sets. The intent of this proposal is to provide budgetary prices for your feasibility study.		
	Package Configuration		
1	i) Trailerized Caterpillar 3412T generator set, rated at 500 KW standby. This unit will be capable of supporting area "A" in the event of a power failure.		
	ii) Trailerized Caterpillar 3406TA/3306TA generator sets, rated at 350 KW and 225 KW respectively. The 3406TA will be capable of supporting selected loads from area B-H. The 3306TA auxiliary generator will provide electrical power for site lighting and non-motor loads.		
	NOTE: The 3406TA/3306TA will be set up for single operation with no provisions for paralleling.		
	Equipment Specification		
	Package #1		
	Caterpillar 3412T		
	Air Cleaners, two-stage, Dry, Panel-Type with Service Indicator		

CROTHERS

YOUR CATERPILLAR DEALER

CROTHERS LIMITED, ONE CROTHERS DRIVE, P.O. BOX 5511, CONCORD, ONTARIO L4K 1E2 (416) 667-5511

QUOTATION

Date October 8, 1986

Page No. Two

QUANTITY	DESCRIPTION	UNIT PRICE	EXTENSION
	Breather, Crankcase		
	Control, Vernier, Right Side (for hydra-mechanical governor only)		
	Cooler, Lubricating Oil, Right Side		
	Exhaust Outlet, flanged, 8 inch		
	Filters		
	Fuel, Right Side		
	Lubricating Oil, Right Side		
	Flywheel		
	Flywheel Housing, SAE No. 0		
	Governor, Hydra-Mechanical (3% Speed Regulation)		
	Instrument Panel, Right Side		
	Fuel Pressure Gauge		
	Lubricating Oil Pressure Gauge		
	Water Temperature Gauge		
	Lifting Eyes		
	Lines, Flexible Fuel		
	Manifolds, Exhaust, Dry		
	Oil Filler in valve cover, Dipstick, both on right side		
	Paint, Caterpillar Yellow		
	Pumps		
	Fuel Priming		
	Fuel Transfer		
	Jacket Water, Gear-Driven, Centrifugal, Right Side		
	SAE Standard Rotation		
	Service Meter		
	Supports		
	Thermostats and Housing		
	Torsional Vibration Damper		
	Variable Timing, Automatic		
	Generator, brushless SR4 with voltage regulator		
	Base, structural steel		
	Radiator/Fan, fan drive		
	Starter, 24 Volt		
	J.W. Heater		
	Miscellaneous Equipment		
	- refurbished 40' trailer		
	- circulation louvers		
	- extended crankcase breathers		
	- extended lube oil drain		

QUOTATION

Date October 8, 1986

QUANTITY	DESCRIPTION	UNIT PRICE	EXTENSION

- Exhaust System
- Battery Rack and Cable
- Fuel tanks (2 X 500 gal.)
- Trailer Markings
- Generator Control Panel
- A.C. Lighting
- D.C. Lighting
- Battery Charger

Package #2

Caterpillar 3406TA/3306TA
(Typical arrangement)

Air Cleaner, Single-Stage, Dry, Panel Type with
 Service Indicator
Base, Structural Steel
Breather, Crankcase
Control Panel, Generator-Mounted, includes:
 Generator Equipment:
  Ammeter, Voltmeter and Switch
  Frequency Meter, Dial Type
  Voltage Adjust Rheostat
 Engine Equipment (24 volt electric):
  Oil Pressure and Water Temperature Gauges
  Manual Start-Stop Control Module
  Engine Control Switch for Off/Reset, Run, Start
  Shutoffs with Single Indicator for Oil Pressure,
   Water Temperature and Overspeed
  Emergency Stop Button
 Panel Features:
  Automatic Start-Stop Control
  Dustproofing,
  Panel Lights
Control, Governor, Vernier, Right Side
Cooler, Lubricating Oil
Elbow, Exhaust, Dry,
Fan (Blower) and Fan Drive
Filters, Fuel and Lubricating Oil

QUOTATION

Date October 8, 1986

QUANTITY	DESCRIPTION	UNIT PRICE	EXTENSION

Flywheel
Gauge, Fuel Pressure
Generator, Brushless SR 4 with voltage regulator
Governor, Hydra-Mechanical (3% speed regulation)
Junction Box with Circuit Breakers for starting system
Lifting Eyes
Lines, Drain (Coolant and Oil)
Lines, Flexible Fuel
Manifold, Exhaust, Dry
Oil Filler in valve cover, dipstick on right side
Paint, Caterpillar Yellow
Pumps
 Fuel Priming
 Fuel Transfer
 Jacket Water, Gear-Driven, Centrifugal
Radiator, with Duct Flange
SAE Standard Rotation
Service Meter
Starting, Electric, 24 volt
Thermostats and Housing
Torsional Vibration Damper
Variable Timing, Fixed
J.W. Heater

Miscellaneous Equipment

- refurbished 40' trailer
- circulation louvers
- extended crankcase breathers
- extended lube oil drain
- exhaust system
- battery, rack and cable
- fuel tanks (2 X 250 ga.)
- trailer markings
- A.C. lighting
- D.C. lighting
- battery charger

CROTHERS
YOUR CATERPILLAR DEALER
CROTHERS LIMITED, ONE CROTHERS DRIVE, P.O. BOX 5511, CONCORD, ONTARIO L4K 1E2 (416) 667-5511

QUOTATION

Date October 8, 1986

Page No. Five

QUANTITY	DESCRIPTION	UNIT PRICE	EXTENSION
	Pricing and Terms		
1	Caterpillar 3412T generator Pkg.	$136,910.	
1	Caterpillar 3406TA/3306TA Generator pkg.	$126,400.	

In Canadian funds, based on 39 Pct. U.S./Cdn. exchange rate and subject to adjustment with exchange rate fluctuation.

Duty included where applicable
Federal Sales Tax extra
Provincial Sales Tax extra
F.O.B. Crothers, Concord
Terms payment, net 30 days

Should any other equipment or services be necessary, please advise. Thank you.

Regards,

CROTHERS LIMITED

Luigi Colangelo

Luigi Colangelo
Estimator
Power Division

CC Perry Fell
Marketing Rep

APPENDIX III

Process Flowsheets

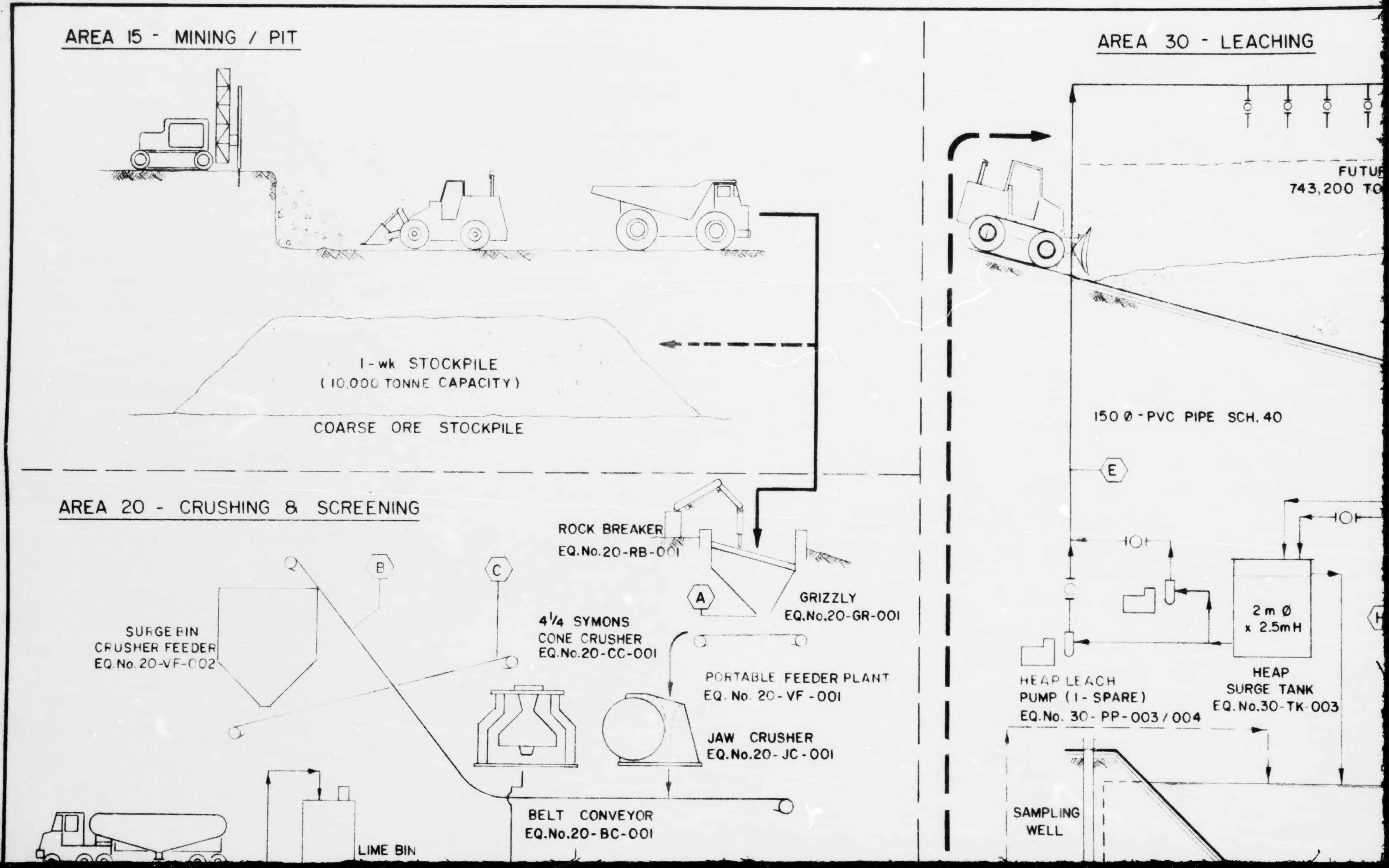
0989A

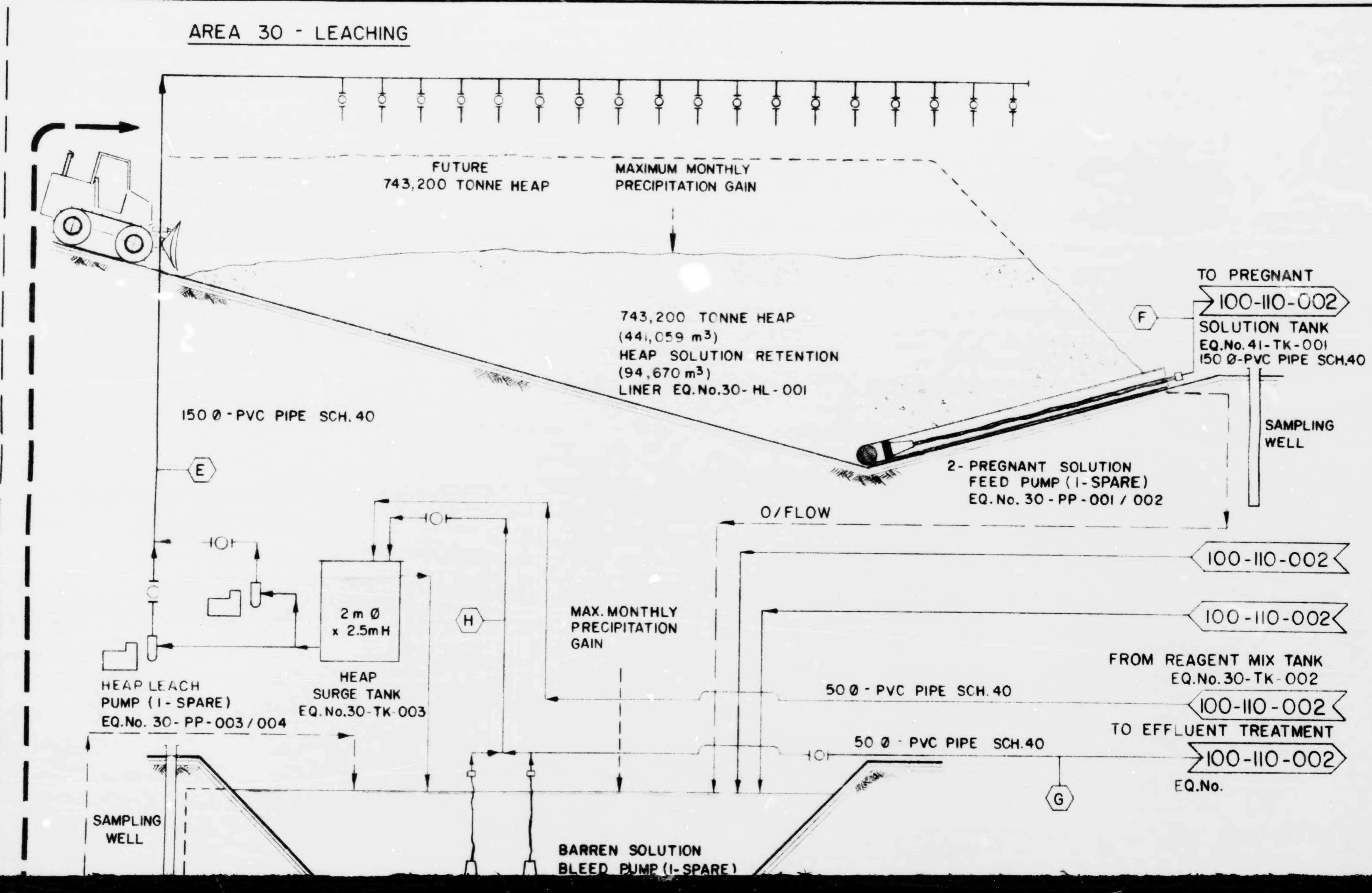
APPENDIX IV

Plant Layout Drawing

APPENDIX V

Site Plan Drawing


AREA 15 - MINING / PIT
AREA 30 - LEACHING
FUTUR
743,200 TO
1-wk STOCKPILE
(10,000 TONNE CAPACITY)
COARSE ORE STOCKPILE
150 Ø - PVC PIPE SCH. 40
E
AREA 20 - CRUSHING & SCREENING
ROCK BREAKER
EQ.No.20-RB-001
B
C
A
GRIZZLY
EQ.No.20-GR-001
SURGE BIN
CRUSHER FEEDER
EQ.No. 20-VF-002
4 1/4 SYMONS
CONE CRUSHER
EQ.No.20-CC-001
PORTABLE FEEDER PLANT
EQ. No. 20-VF-001
JAW CRUSHER
EQ.No.20-JC-001
2 m Ø
x 2.5mH
H
HEAP LEACH
PUMP (1-SPARE)
EQ.No. 30-PP-003/004
HEAP
SURGE TANK
EQ. No.30-TK-003
SAMPLING
WELL
BELT CONVEYOR
EQ.No.20-BC-001
LIME BIN


AREA 30 - LEACHING
FUTURE
743,200 TONNE HEAP
MAXIMUM MONTHLY
PRECIPITATION GAIN
TO PREGNANT
100-110-002
F
SOLUTION TANK
EQ.No. 41-TK-001
150 Ø-PVC PIPE SCH.40
743,200 TONNE HEAP
(441,059 m3)
HEAP SOLUTION RETENTION
(94,670 m3)
LINER EQ.No.30-HL-001
SAMPLING
WELL
150 Ø-PVC PIPE SCH. 40
E
2- PREGNANT SOLUTION
FEED PUMP (1-SPARE)
EQ. No. 30-PP-001 / 002
O/FLOW
100-110-002
100-110-002
2 m Ø
x 2.5mH
H
MAX. MONTHLY
PRECIPITATION
GAIN
FROM REAGENT MIX TANK
EQ.No. 30-TK-002
HEAP LEACH
PUMP (1-SPARE)
EQ.No. 30-PP-003/004
HEAP
SURGE TANK
EQ. No.30-TK-003
50 Ø - PVC PIPE SCH. 40
100-110-002
TO EFFLUENT TREATMENT
50 Ø - PVC PIPE SCH.40
100-110-002
EQ.No.
G
SAMPLING
WELL
BARREN SOLUTION
BLEED PUMP (1-SPARE)
FEED PUMP (1-SPARE)

NOTES

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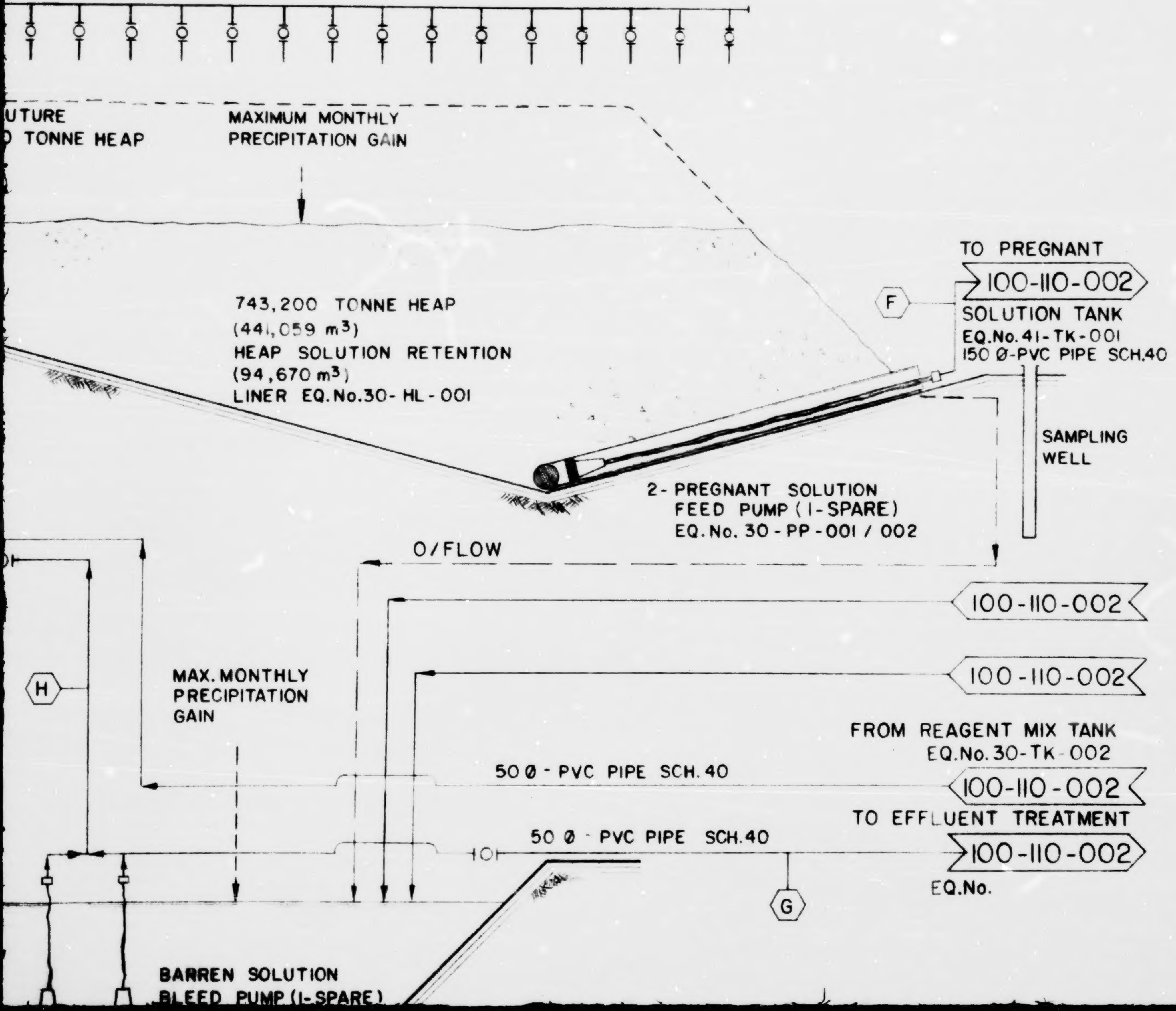
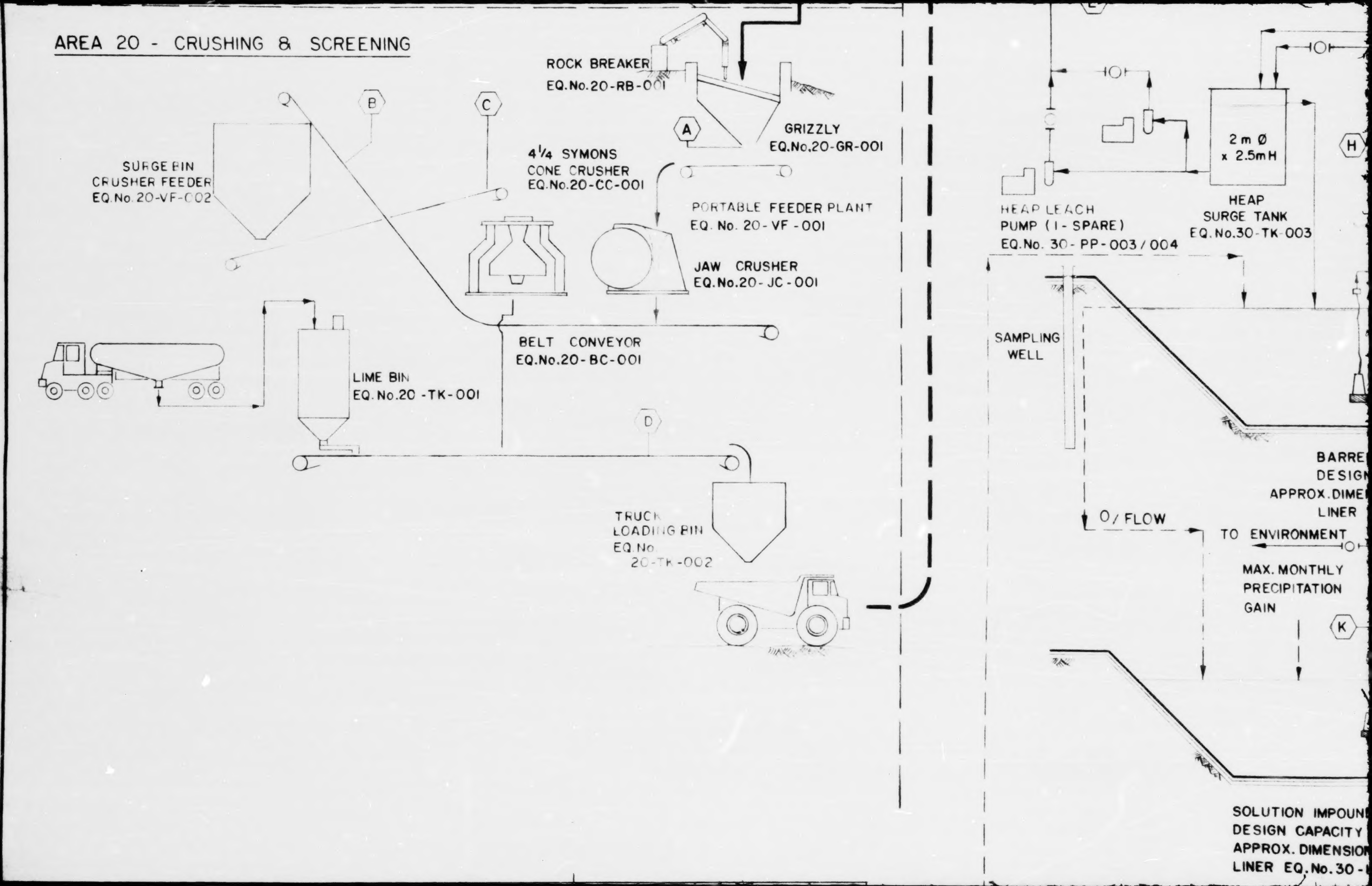

UTURE
) TONNE HEAP
MAXIMUM MONTHLY
PRECIPITATION GAIN
743,200 TONNE HEAP
(441,059 m³)
HEAP SOLUTION RETENTION
(94,670 m³)
LINER EQ.No.30-HL-001
TO PREGNANT
100-110-002
SOLUTION TANK
EQ.No.41-TK-001
150 Ø-PVC PIPE SCH.40
F
SAMPLING
WELL
2- PREGNANT SOLUTION
FEED PUMP (1-SPARE)
EQ.No. 30-PP-001/002
O/FLOW
100-110-002
H
MAX. MONTHLY
PRECIPITATION
GAIN
100-110-002
FROM REAGENT MIX TANK
EQ.No.30-TK-002
50 Ø - PVC PIPE SCH.40
100-110-002
TO EFFLUENT TREATMENT
50 Ø - PVC PIPE SCH.40
100-110-002
EQ.No.
G
BARREN SOLUTION
BLEED PUMP (1-SPARE)

NOTES:

1. ALL MINING AND ORE HANDLING TO COARSE ORE STOCKPILE, LOCATED ADJACENT TO THE CRUSHING PLANT WILL BE ACCOMPLISHED BY AN INDEPENDENT CONTRACTOR.
2. THE ORE HEAD GRADE IS ASSUMED TO BE 1.14 GRAMS PER TONNE (.033 oz/ton)
3. ORE HANDLING FROM THE COARSE ORE STOCKPILE TO THE CRUSHING PLANT AND CRUSHING PLANT OPERATION WILL BE BY OWNER. CRUSHED ORE HAULAGE TO HEAP BY INDEPENDENT CONTRACTOR.
4. CRUSHING PLANT FEED WILL BE -600mm ORE. CRUSHING PLANT PRODUCT WILL BE 100% -35 mm ORE.
5. INITIAL HEAP WILL BE 743,200 TONNES (441,059 m³)
6. THE ASSUMED ANGLE OF REPOSE FOR IS 35°. MAXIMUM HEIGHT OF SECOND HEAP LIFT WILL BE APPROXIMATELY 76 m.
7. HEAP PERCOLATION RATE = .1019 l/min/m² (0.0025 US GPM/FT²)
8. HEAP LINER SLOPE IS VARIABLE.
9. HEAP LINER CONSTRUCTION: BEDROCK, FINE SAND, 60 MIL HIGH DENSITY POLYETHYLENE, SAND, GRAVEL AND -35mm ORE.
10. POND LINER CONSTRUCTION: BEDROCK, FINE SAND AND 60 MIL HIGH DENSITY POLYETHYLENE.
11. POND WALL SLOPE = 2:1
12. MAX. MONTHLY PRECIPITATION = 56.7mm (JULY) (DECEMBER) TOTAL COLLECTION AREA
13. MINIMUM HEAP LEACHING TIME = 90 DAYS PER CYCLE.
14. DESIGN LEACH RECOVERY = 75% IN 90 DAYS
15. LEACHING OPERATION (PUMPS): 24 HRS PER DAY, 7 DAYS PER WEEK. 180 DAYS PER YEAR.
16. ALL EQUIPMENT SIZED FOR 743,200 TONNES ORE IN 300 DAYS (2477 TONNES PER DAY)


AREA 20 - CRUSHING & SCREENING
ROCK BREAKER
EQ.No.20-RB-001
GRIZZLY
EQ.No.20-GR-001
A
B
C
D
SURGE BIN
CRUSHER FEEDER
EQ.No.20-VF-002
4¼ SYMONS
CONE CRUSHER
EQ.No.20-CC-001
PORTABLE FEEDER PLANT
EQ. No. 20-VF-001
JAW CRUSHER
EQ.No.20-JC-001
BELT CONVEYOR
EQ.No.20-BC-001
LIME BIN
EQ.No.20-TK-001
TRUCK
LOADING BIN
EQ.No
20-TK-002
HEAP LEACH
PUMP (1-SPARE)
EQ.No. 30-PP-003/004
2 m Ø
x 2.5mH
HEAP
SURGE TANK
EQ.No.30-TK-003
H
SAMPLING
WELL
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DESIGN
APPROX.DIME
LINER
O/FLOW
TO ENVIRONMENT
MAX. MONTHLY
PRECIPITATION
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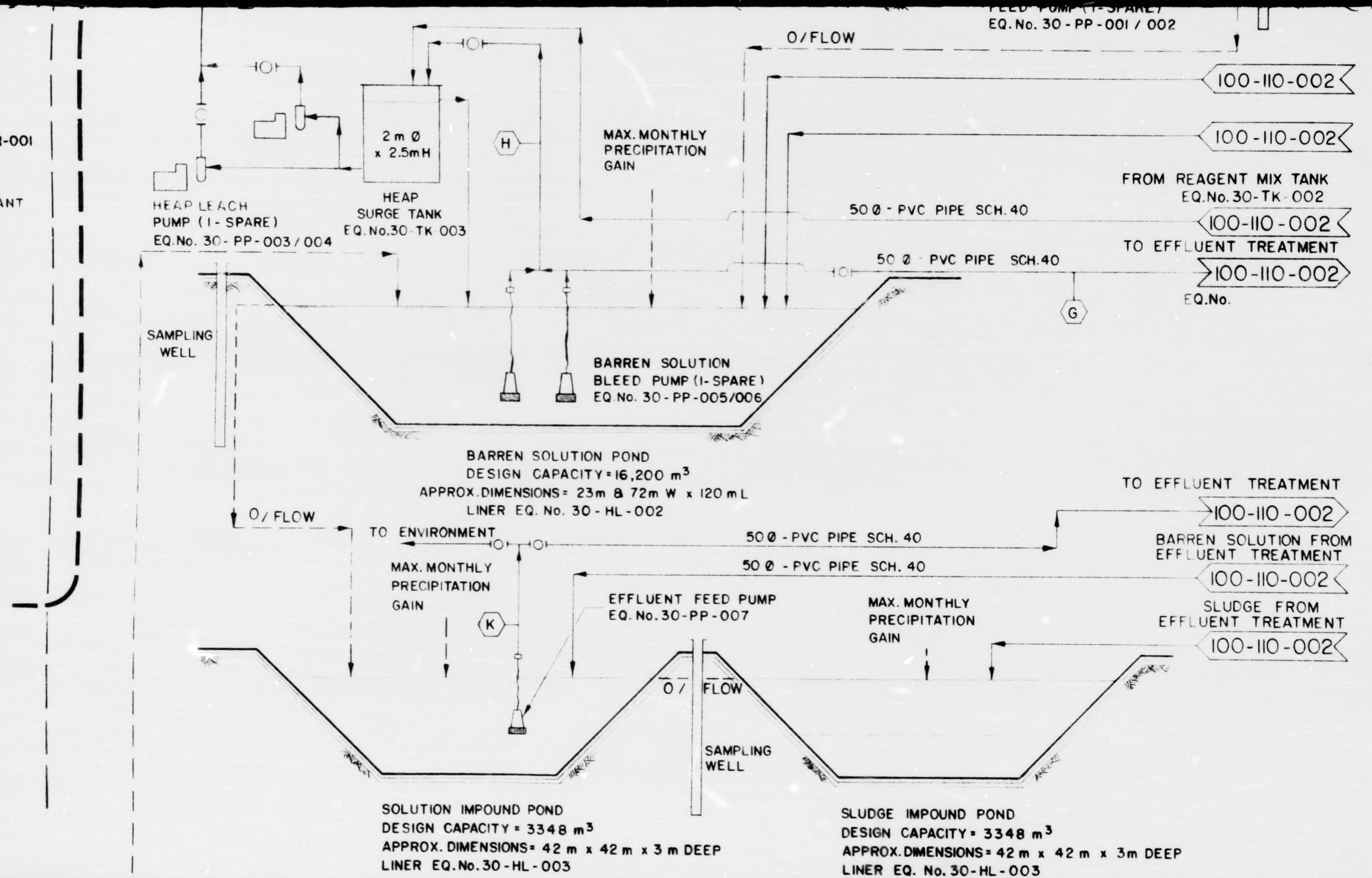
SOLUTION IMPOUN
DESIGN CAPACITY
APPROX. DIMENSIO
LINER EQ.No.30-


-001
ANT
FEED PUMP (1-SPARE)
EQ. No. 30 - PP - 001 / 002
O/FLOW
100-110-002
100-110-002
2 m Ø
x 2.5mH
H
MAX. MONTHLY
PRECIPITATION
GAIN
HEAP LEACH
PUMP (1 - SPARE)
EQ. No. 30 - PP - 003 / 004
HEAP
SURGE TANK
EQ. No. 30 - TK 003
FROM REAGENT MIX TANK
EQ. No. 30 - TK - 002
50 Ø - PVC PIPE SCH. 40
100-110-002
TO EFFLUENT TREATMENT
50 Ø - PVC PIPE SCH. 40
100-110-002
EQ. No.
G
SAMPLING
WELL
BARREN SOLUTION
BLEED PUMP (1- SPARE)
EQ. No. 30 - PP - 005/006
BARREN SOLUTION POND
DESIGN CAPACITY = 16,200 m³
APPROX. DIMENSIONS = 23m & 72m W x 120 m L
LINER EQ. No. 30 - HL - 002
TO EFFLUENT TREATMENT
100-110-002
O/FLOW
TO ENVIRONMENT
50 Ø - PVC PIPE SCH. 40
BARREN SOLUTION FROM
EFFLUENT TREATMENT
50 Ø - PVC PIPE SCH. 40
100-110-002
MAX. MONTHLY
PRECIPITATION
GAIN
EFFLUENT FEED PUMP
EQ. No. 30 - PP - 007
K
MAX. MONTHLY
PRECIPITATION
GAIN
SLUDGE FROM
EFFLUENT TREATMENT
100-110-002
O/ FLOW
SAMPLING
WELL
SOLUTION IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3 m DEEP
LINER EQ. No. 30 - HL - 003
SLUDGE IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3m DEEP
LINER EQ. No. 30 - HL - 003

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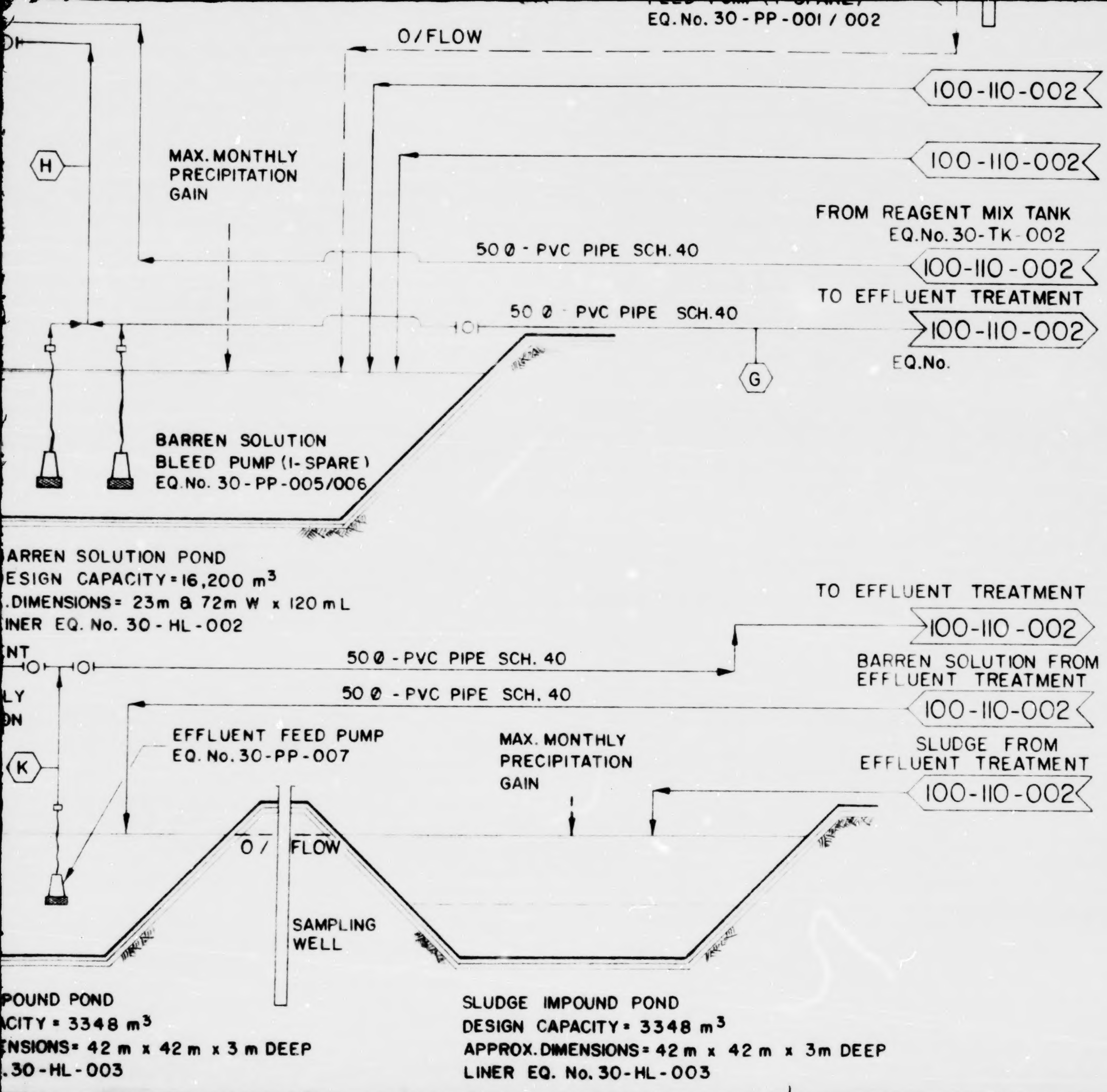
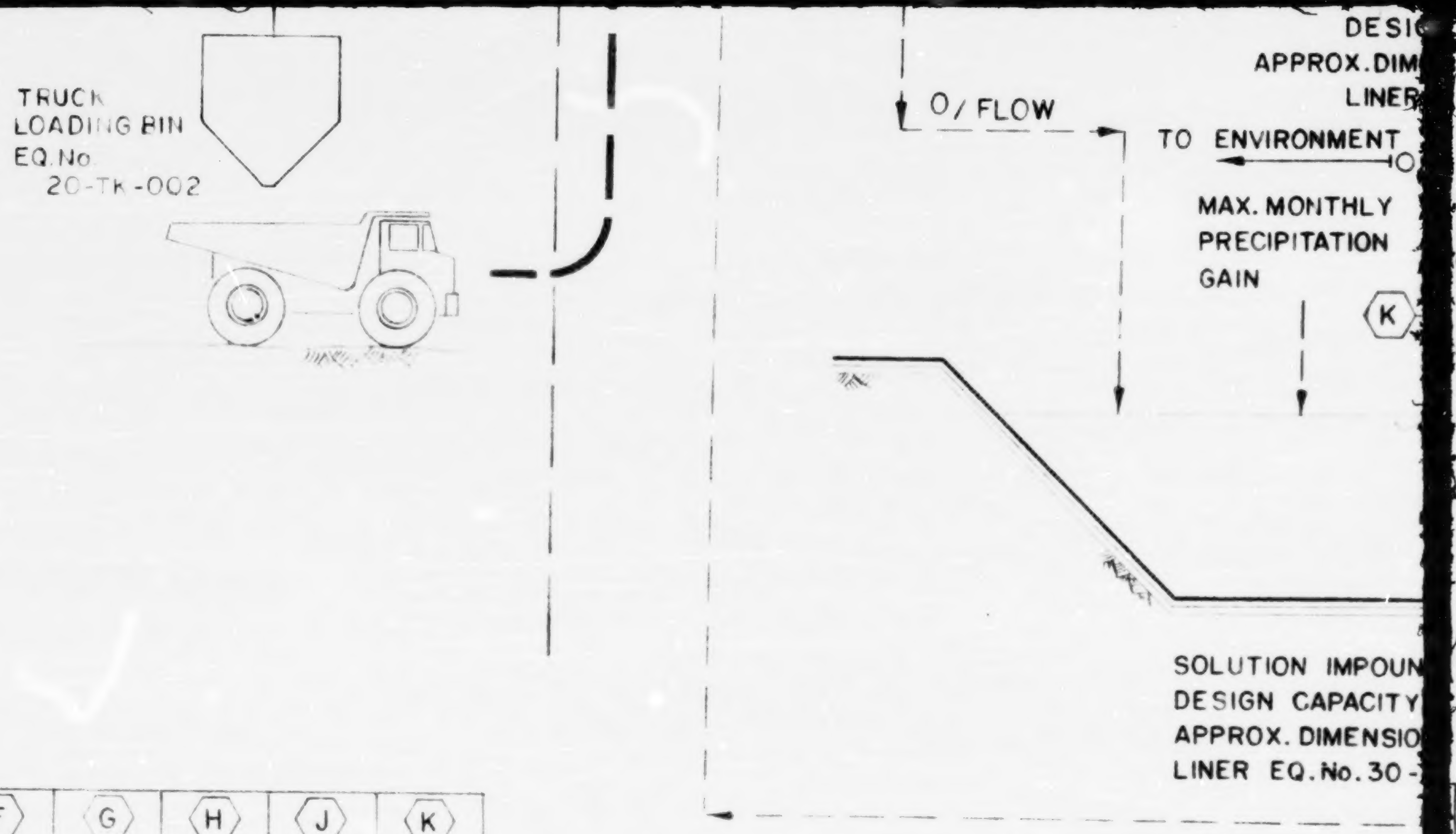

EQ. No. 30 - PP - 001 / 002
O/FLOW
100-110-002
100-110-002
H
MAX. MONTHLY PRECIPITATION GAIN
FROM REAGENT MIX TANK EQ.No. 30-TK-002
50 Ø - PVC PIPE SCH. 40
100-110-002
TO EFFLUENT TREATMENT
50 Ø PVC PIPE SCH.40
100-110-002
EQ.No.
G
BARREN SOLUTION BLEED PUMP (1-SPARE) EQ.No. 30-PP-005/006
ARREN SOLUTION POND
ESIGN CAPACITY = 16,200 m³
.DIMENSIONS = 23m & 72m W x 120 m L
INER EQ. No. 30 - HL - 002
TO EFFLUENT TREATMENT
100-110-002
NT
50 Ø - PVC PIPE SCH. 40
BARREN SOLUTION FROM EFFLUENT TREATMENT
LY
ON
50 Ø - PVC PIPE SCH. 40
100-110-002
EFFLUENT FEED PUMP EQ. No. 30-PP-007
K
MAX. MONTHLY PRECIPITATION GAIN
SLUDGE FROM EFFLUENT TREATMENT
100-110-002
O / FLOW
SAMPLING WELL
POUND POND
ACITY = 3348 m³
ENSIONS = 42 m x 42 m x 3 m DEEP
.30-HL-003
SLUDGE IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3m DEEP
LINER EQ. No. 30-HL-003

LIFT WILL BE APPROXIMATELY 76 m.

7. HEAP PERCOLATION RATE = .1019 l/min/m² (0.0025 US GPM/FT²)
8. HEAP LINER SLOPE IS VARIABLE.
9. HEAP LINER CONSTRUCTION: BEDROCK, FINE SAND, 60 MIL HIGH DENSITY POLYETHYLENE, SAND, GRAVEL AND -35mm ORE.
10. POND LINER CONSTRUCTION: BEDROCK, FINE SAND AND 60 MIL HIGH DENSITY POLYETHYLENE.
11. POND WALL SLOPE = 2:1
12. MAX. MONTHLY PRECIPITATION = 56.7mm (JULY) (DECEMBER) TOTAL COLLECTION AREA
13. MINIMUM HEAP LEACHING TIME = 90 DAYS PER CYCLE
14. DESIGN LEACH RECOVERY = 75 % IN 90 DAYS
15. LEACHING OPERATION (PUMPS): 24 HRS PER DAY, 7 DAYS PER WEEK. 180 DAYS PER YEAR.
16. ALL EQUIPMENT SIZED FOR 743,200 TONNES ORE IN 300 DAYS (2477 TONNES PER DAY).
17. FLOW RATES AS SHOWN ARE FOR 743,200 TONNE PER ANNUM HEAP.
18. Na CN CONSUMPTION = 2.0 kg / tonne OF ORE. LIME CONSUMPTION 4.0 kg / tonne OF ORE.

LEGEND:

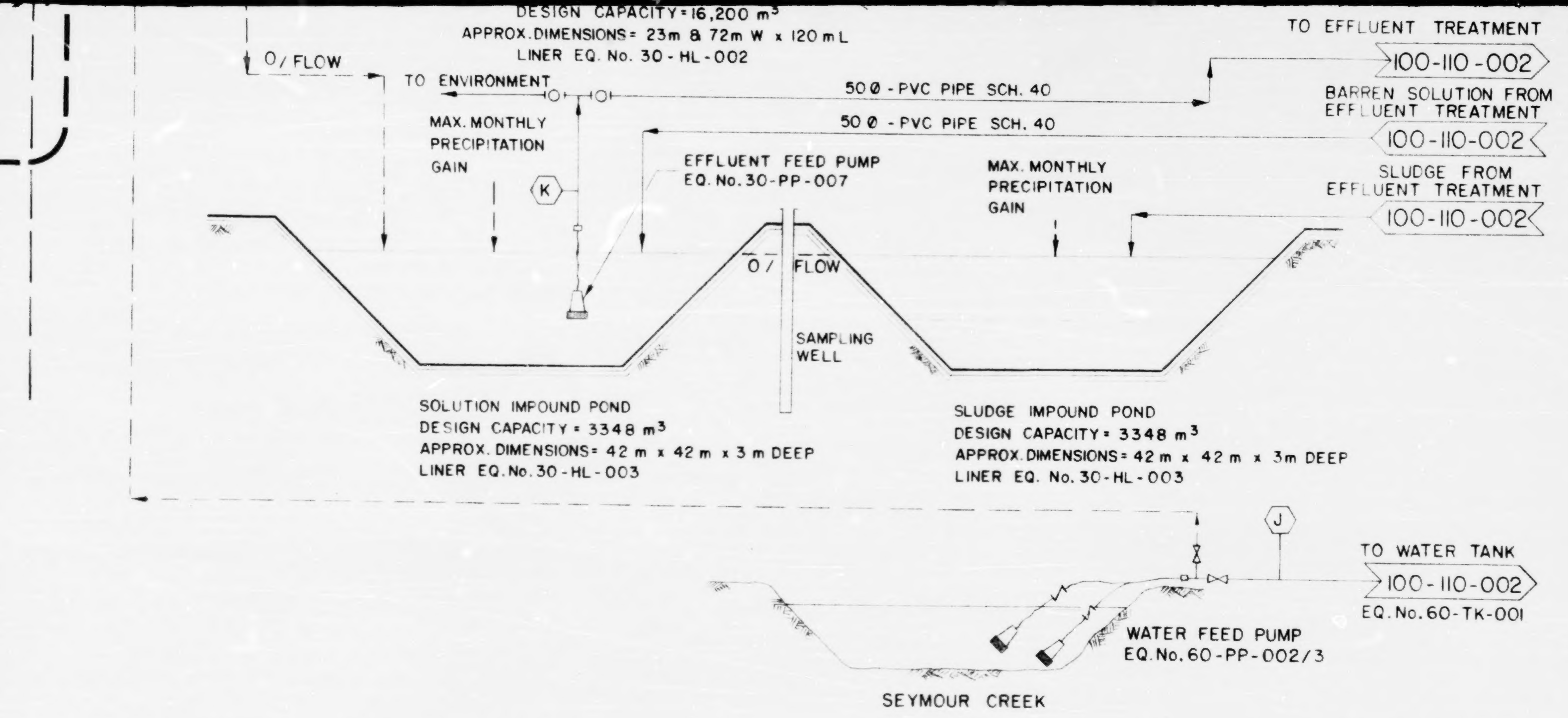
ITEM	DESCRIPTION


TRUCK
LOADING BIN
EQ.No.
20-TK-002
DESIG
APPROX.DIM
LINER
O/FLOW
TO ENVIRONMENT
MAX. MONTHLY
PRECIPITATION
GAIN
K
SOLUTION IMPOUN
DESIGN CAPACITY
APPROX. DIMENSIO
LINER EQ.No.30-

VARIABLES / STREAM	A	B	C	D	E	F	G	H	J	K
DRY TONNES SOLIDS/HR	172			172	–	–	–	–	–	–
BULK SPECIFIC WEIGHT kg/m³	1682	1682	1682	1682	–	–	–	–	–	–
LIQUID FLOW liters/second	–	–	–	–	20.2	22.1	3.33	20.17	1.20	3.56
% SOLIDS BY WEIGHT	100	100	100	100	0	0	0	0	0	0
GOLD grams/tonne	1.13	1.13	1.13	1.13	.005	2.87	.005	.005	–	–
GOLD ppm	–	–	–	–	.005	2.87	.005	.005	–	–
SPECIFIC GRAVITY	2.75	2.75	2.75	2.75	1.0	1.0	1.0	1.0	1.0	1.0
ph OF SOLUTION	–	–	–	–	11.0	11.0	11.0	11.0	7.0	10.0

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHEC
40-130-002	PROCESS PLANT LAYOUT – SECTION							
40-130-001	PROCESS PLANT LAYOUT – PLAN							
10-120-002	LOCAL SITE LAYOUT							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND							
30-130-001	HEAP GENERAL LAYOUT							
10-120-001	GENERAL SITE PLAN							
10-110-002	FLOW DIAGRAM (SHEET 2 OF 2)							

TRANSTEX


DESIGN CAPACITY = 16,200 m³
APPROX. DIMENSIONS = 23m & 72m W x 120 m L
LINER EQ. No. 30 - HL - 002
O/ FLOW
TO ENVIRONMENT
TO EFFLUENT TREATMENT
100-110-002
50 Ø - PVC PIPE SCH. 40
BARREN SOLUTION FROM EFFLUENT TREATMENT
50 Ø - PVC PIPE SCH. 40
100-110-002
MAX. MONTHLY PRECIPITATION GAIN
K
EFFLUENT FEED PUMP
EQ. No. 30-PP-007
MAX. MONTHLY PRECIPITATION GAIN
SLUDGE FROM EFFLUENT TREATMENT
100-110-002
O/ FLOW
SAMPLING WELL
SOLUTION IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3 m DEEP
LINER EQ. No. 30 - HL - 003
SLUDGE IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3m DEEP
LINER EQ. No. 30 - HL - 003
J
TO WATER TANK
100-110-002
EQ. No. 60-TK-001
WATER FEED PUMP
EQ. No. 60-PP-002/3
SEYMOUR CREEK

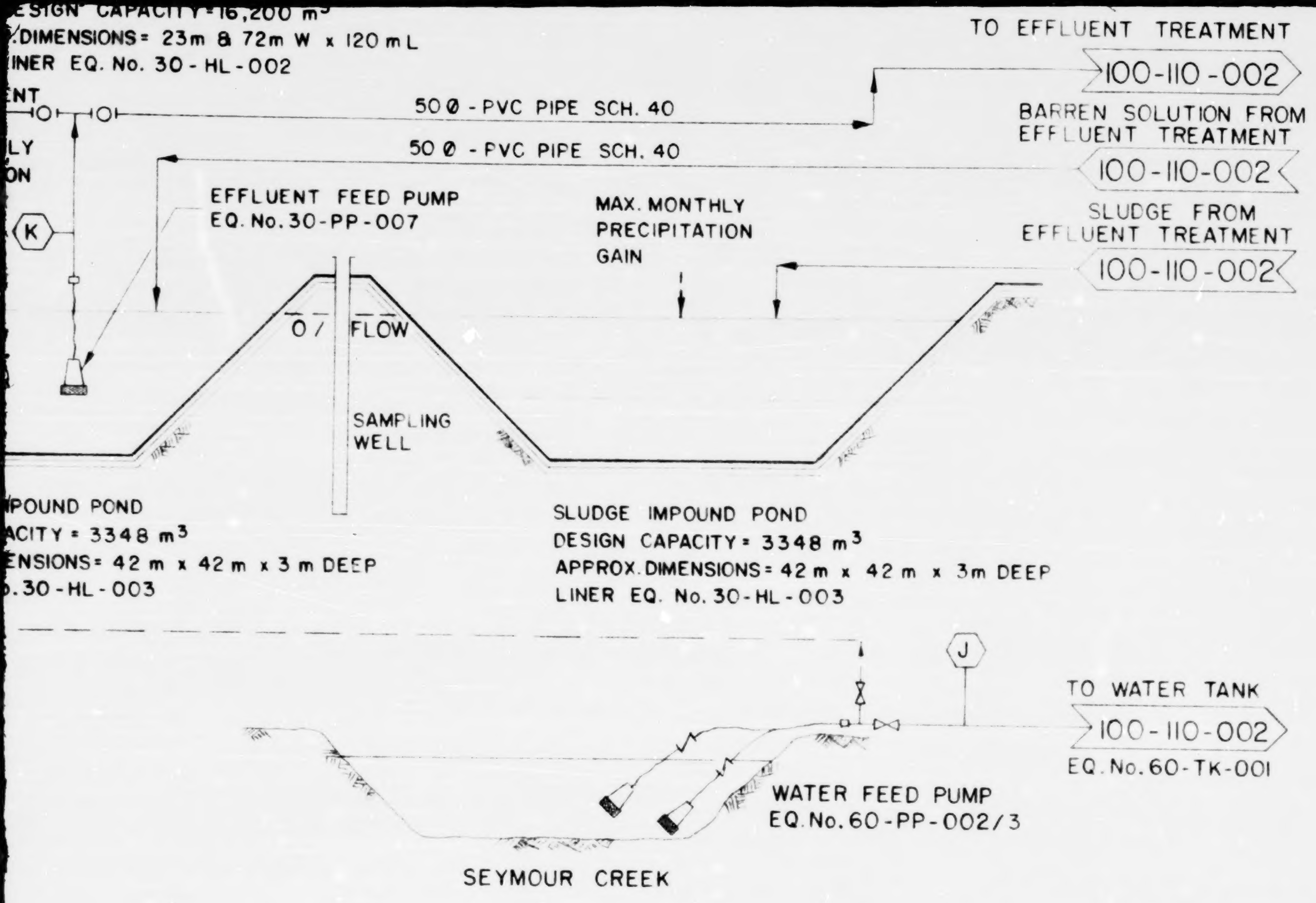
No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK			
										SCALE	NTS	
										DRAWN	E.N.	NOV. 14,86
										CHECKED		
										APPROVAL		
					P2	MAY 20,87	RELEASED FOR REPORT	J. LI				
					P1		RELEASED FOR INFORMATION	E.N.				

WDI

DSS

FLOW

DATE


ESIGN CAPACITY=16,200 m³
.DIMENSIONS= 23m & 72m W x 120 m L
INER EQ. No. 30 - HL - 002
ENT
LY
ON
K
TO EFFLUENT TREATMENT
100-110-002
50 0 - PVC PIPE SCH. 40
BARREN SOLUTION FROM EFFLUENT TREATMENT
100-110-002
50 0 - PVC PIPE SCH. 40
EFFLUENT FEED PUMP EQ. No. 30-PP-007
MAX. MONTHLY PRECIPITATION GAIN
SLUDGE FROM EFFLUENT TREATMENT
100-110-002
O/ FLOW
SAMPLING WELL
POUND POND
ACITY = 3348 m³
ENSIONS= 42 m x 42 m x 3 m DEEP
.30 - HL - 003
SLUDGE IMPOUND POND
DESIGN CAPACITY = 3348 m³
APPROX. DIMENSIONS = 42 m x 42 m x 3m DEEP
LINER EQ. No. 30 - HL - 003
J
TO WATER TANK
100-110-002
EQ. No. 60-TK-001
WATER FEED PUMP EQ. No. 60-PP-002/3
SEYMOUR CREEK

LEGEND:

ITEM	DESCRIPTION
———	CONTINUOUS PROCESS FLOW
- - - -	FUTURE OR INTERMITTENT FLOW
—⋈—	GATE VALVE
—IOI—	BALL VALVE
—|\\\|—	CHECK VALVE
—|/|—	BUTTERFLY VALVE

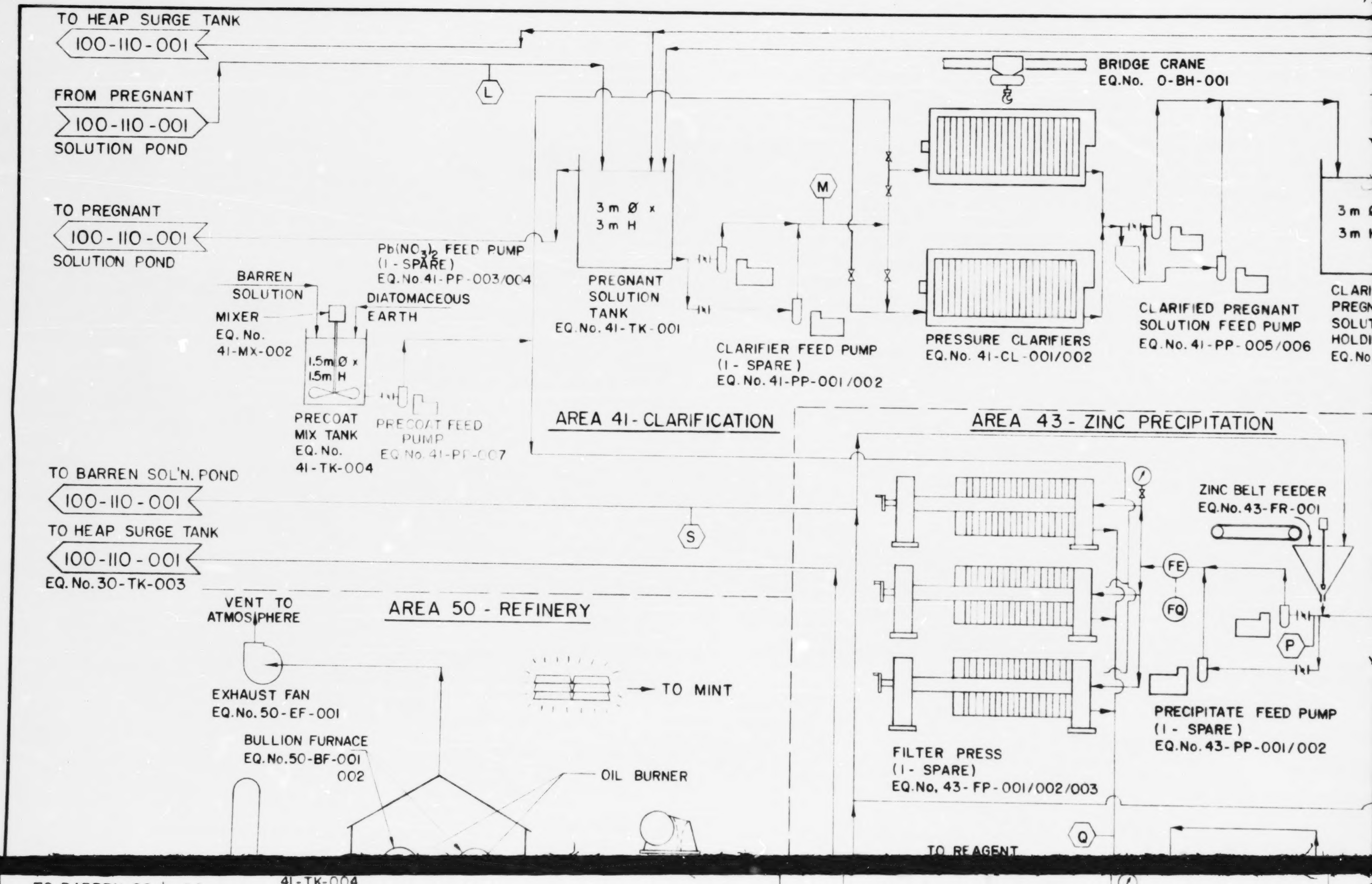
CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
	P1		RELEASED FOR INFORMATION	E.N.	

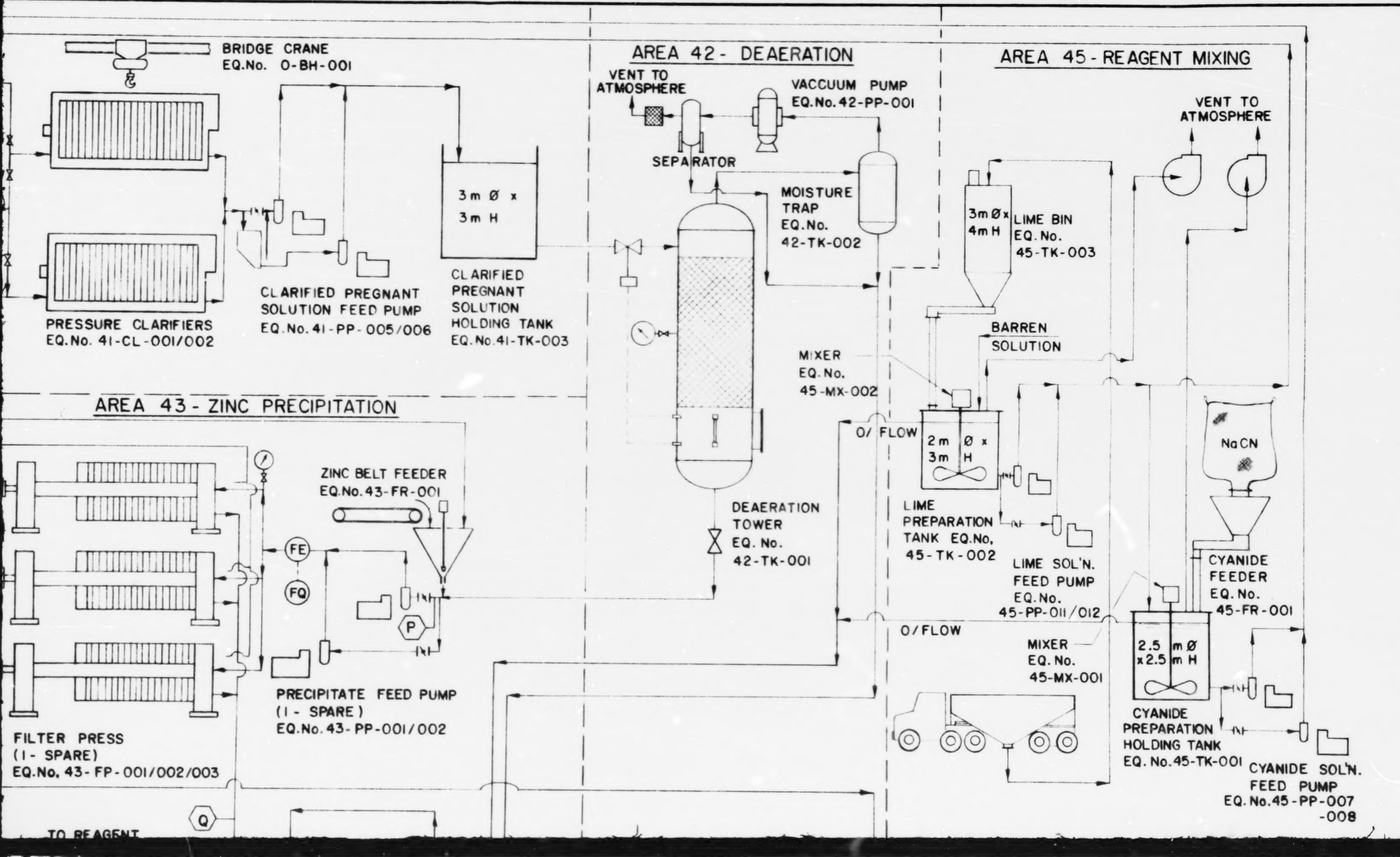
SCALE	NTS	
DRAWN	E.N.	NOV. 14,86
CHECKED		
APPROVAL		

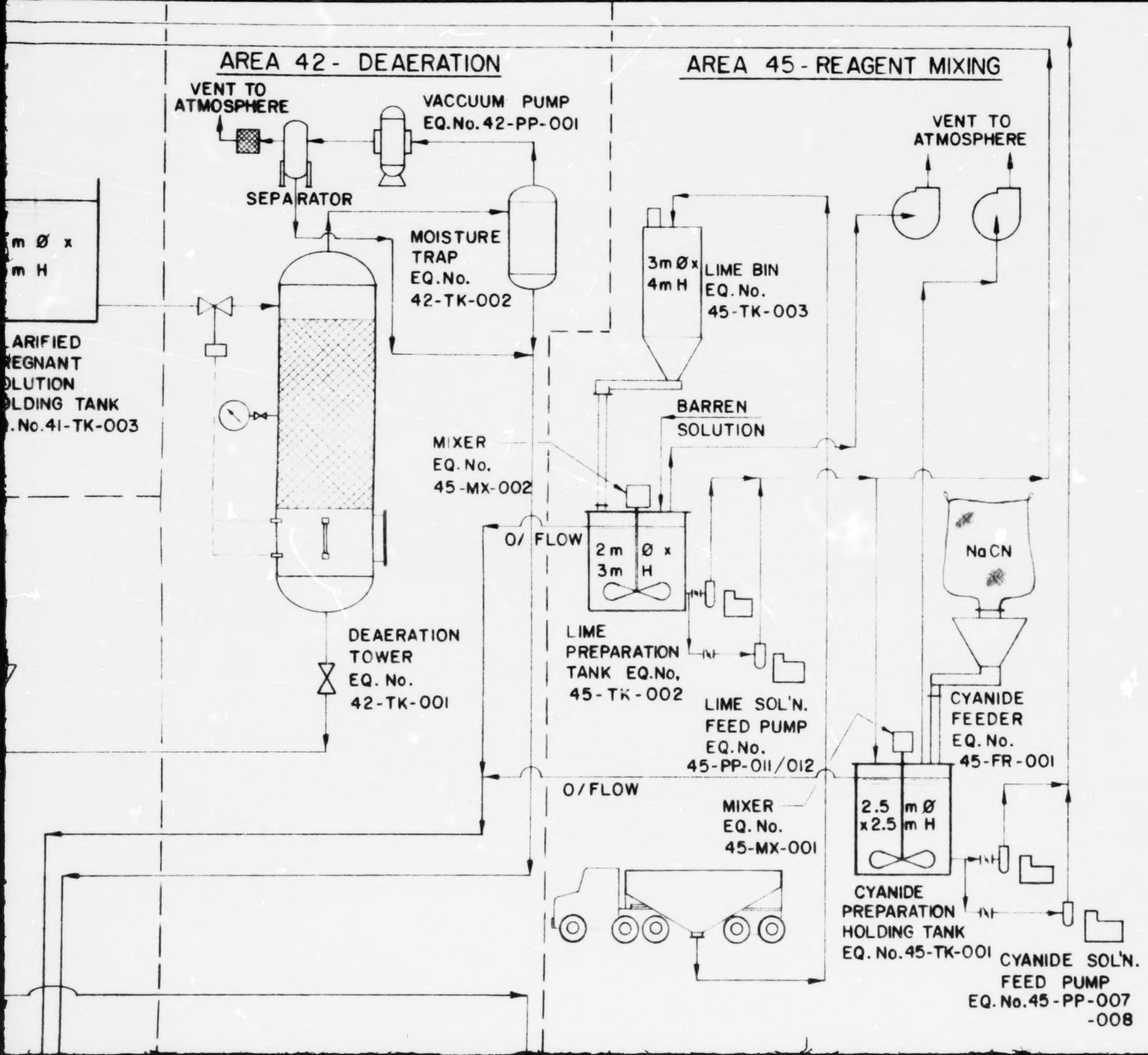
WITTECK DEVELOPMENT INC.

DSS - YUKON HEAP LEACH PROJECT
GENERAL (AREA No. 10)
HEAP LEACHING PROCESS
FLOW DIAGRAM (SHEET 1 OF 2)

DATE	PROJECT NO.	DWG. No	REV.
	5209	10 - 110 - 001	P2


TO HEAP SURGE TANK
100-110-001
FROM PREGNANT
100-110-001
SOLUTION POND
TO PREGNANT
100-110-001
SOLUTION POND
L
BRIDGE CRANE
EQ.No. 0-BH-001
M
3 m Ø x
3 m H
PREGNANT
SOLUTION
TANK
EQ.No. 41-TK-001
Pb(NO3)2 FEED PUMP
(1 - SPARE)
EQ.No.41-PP-003/004
BARREN
SOLUTION
MIXER
EQ. No.
41-MX-002
DIATOMACEOUS
EARTH
1.5m Ø x
1.5m H
PRECOAT
MIX TANK
EQ. No.
41-TK-004
PRECOAT FEED
PUMP
EQ.No. 41-PP-007
CLARIFIER FEED PUMP
(1 - SPARE)
EQ. No. 41-PP-001/002
PRESSURE CLARIFIERS
EQ.No. 41-CL-001/002
CLARIFIED PREGNANT
SOLUTION FEED PUMP
EQ.No. 41-PP-005/006
3 m
3 m
CLARI
PREGN
SOLUT
HOLDI
EQ.No
AREA 41 - CLARIFICATION
AREA 43 - ZINC PRECIPITATION
TO BARREN SOL'N. POND
100-110-001
S
TO HEAP SURGE TANK
100-110-001
EQ. No. 30-TK-003
ZINC BELT FEEDER
EQ.No. 43-FR-001
FE
FQ
P
PRECIPITATE FEED PUMP
(1 - SPARE)
EQ.No. 43-PP-001/002
FILTER PRESS
(1 - SPARE)
EQ.No. 43-FP-001/002/003
VENT TO
ATMOSPHERE
AREA 50 - REFINERY
EXHAUST FAN
EQ.No. 50-EF-001
TO MINT
BULLION FURNACE
EQ.No. 50-BF-001
002
OIL BURNER
Q
TO REAGENT
41-TK-004


BRIDGE CRANE
EQ.No. 0-BH-001
PRESSURE CLARIFIERS
EQ.No. 41-CL-001/002
CLARIFIED PREGNANT
SOLUTION FEED PUMP
EQ.No. 41-PP-005/006
3m Ø x
3m H
CLARIFIED
PREGNANT
SOLUTION
HOLDING TANK
EQ.No. 41-TK-003
AREA 42 - DEAERATION
VENT TO
ATMOSPHERE
SEPARATOR
VACCUUM PUMP
EQ.No. 42-PP-001
MOISTURE
TRAP
EQ.No.
42-TK-002
DEAERATION
TOWER
EQ. No.
42-TK-001
AREA 45 - REAGENT MIXING
VENT TO
ATMOSPHERE
3m Ø x
4m H
LIME BIN
EQ. No.
45-TK-003
BARREN
SOLUTION
MIXER
EQ. No.
45-MX-002
O/ FLOW
2m Ø x
3m H
LIME
PREPARATION
TANK EQ.No,
45-TK-002
LIME SOL'N.
FEED PUMP
EQ. No.
45-PP-011/012
NaCN
CYANIDE
FEEDER
EQ. No.
45-FR-001
O/FLOW
MIXER
EQ. No.
45-MX-001
2.5 m Ø
x 2.5 m H
CYANIDE
PREPARATION
HOLDING TANK
EQ. No. 45-TK-001
CYANIDE SOL'N.
FEED PUMP
EQ. No. 45-PP-007
-008
AREA 43 - ZINC PRECIPITATION
ZINC BELT FEEDER
EQ.No. 43-FR-001
FE
FQ
P
PRECIPITATE FEED PUMP
(1 - SPARE)
EQ.No. 43-PP-001/002
FILTER PRESS
(1 - SPARE)
EQ.No. 43-FP-001/002/003
Q
TO REAGENT


AREA 42 - DEAERATION
AREA 45 - REAGENT MIXING
VENT TO ATMOSPHERE
VACCUUM PUMP EQ.No. 42-PP-001
SEPARATOR
MOISTURE TRAP EQ.No. 42-TK-002
m Ø x
m H
LARIFIED
REGNANT
OLUTION
OLDING TANK
.No.41-TK-003
DEAERATION TOWER EQ. No. 42-TK-001
O/ FLOW
3m Ø x 4m H
LIME BIN EQ. No. 45-TK-003
BARREN SOLUTION
MIXER EQ. No. 45-MX-002
2m Ø x 3m H
LIME PREPARATION TANK EQ.No. 45-TK-002
LIME SOL'N. FEED PUMP EQ. No. 45-PP-011/012
VENT TO ATMOSPHERE
Na CN
CYANIDE FEEDER EQ. No. 45-FR-001
O/FLOW
MIXER EQ. No. 45-MX-001
2.5 m Ø x 2.5 m H
CYANIDE PREPARATION HOLDING TANK EQ. No. 45-TK-001
CYANIDE SOL'N. FEED PUMP EQ. No. 45-PP-007 -008

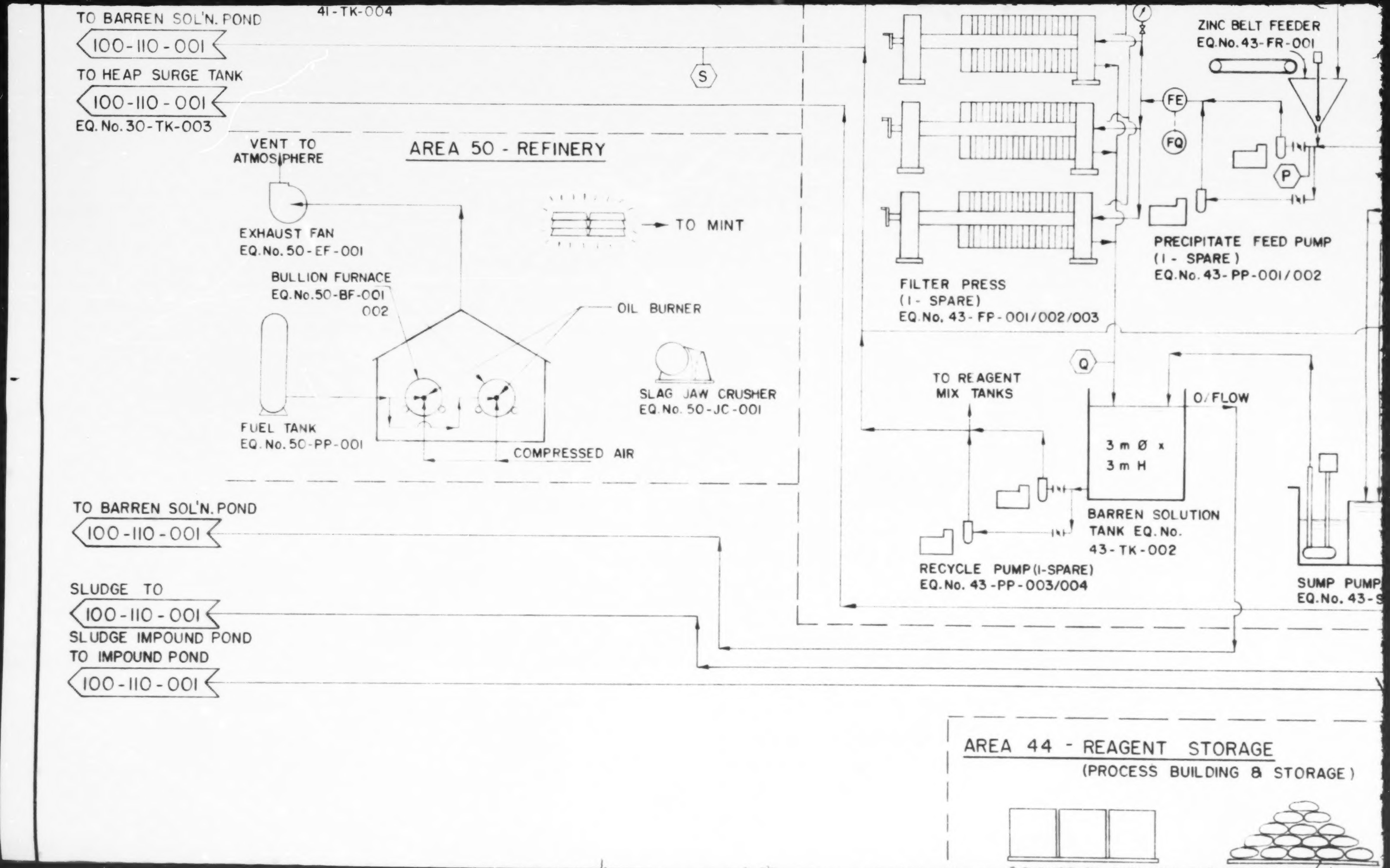
NOTES:

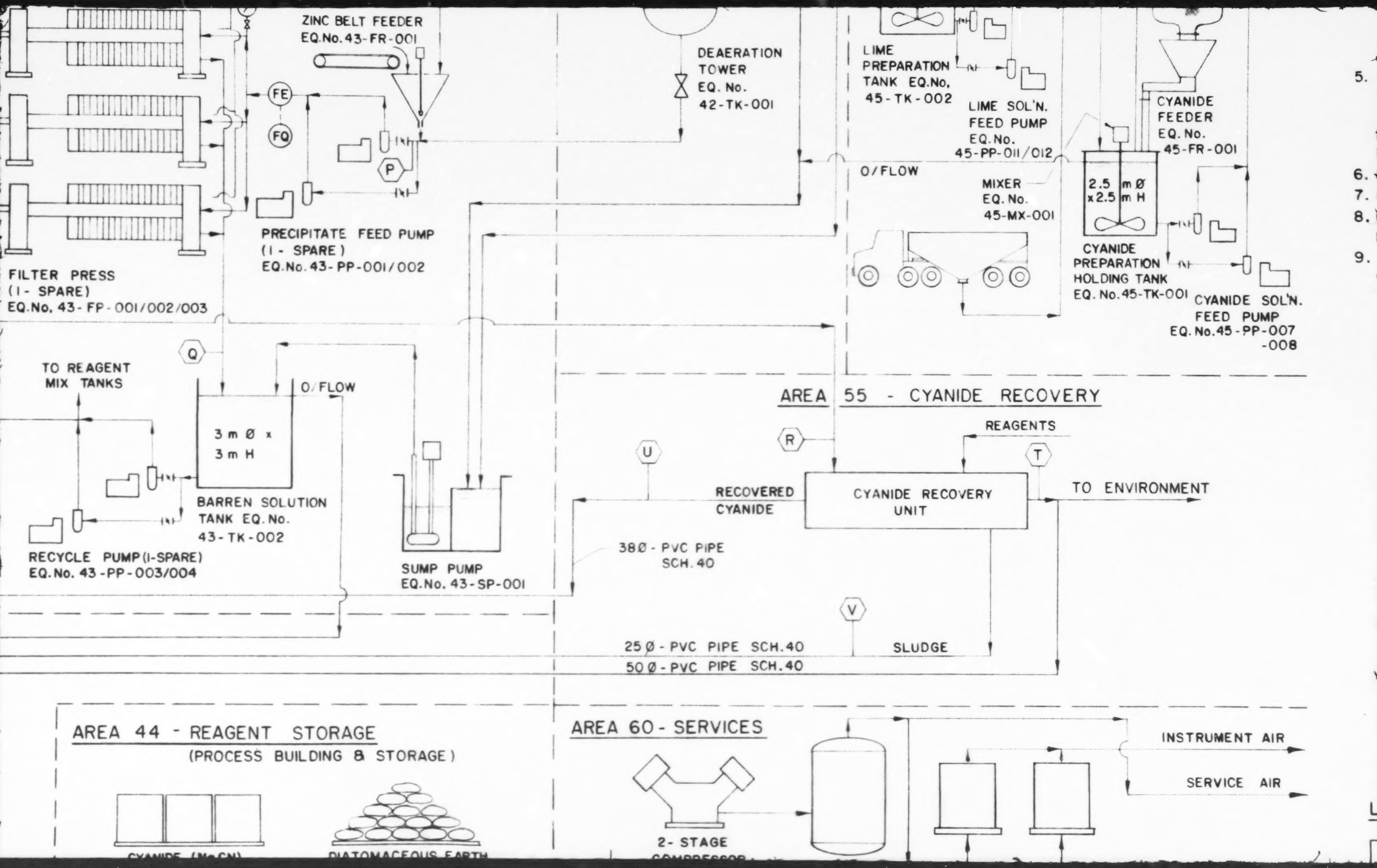
1. ALL EQUIPMENT IS SIZED FOR 743,200 TONNES OF ORE LEACHED FOR 90 DAYS AT 75 % RECOVERY.

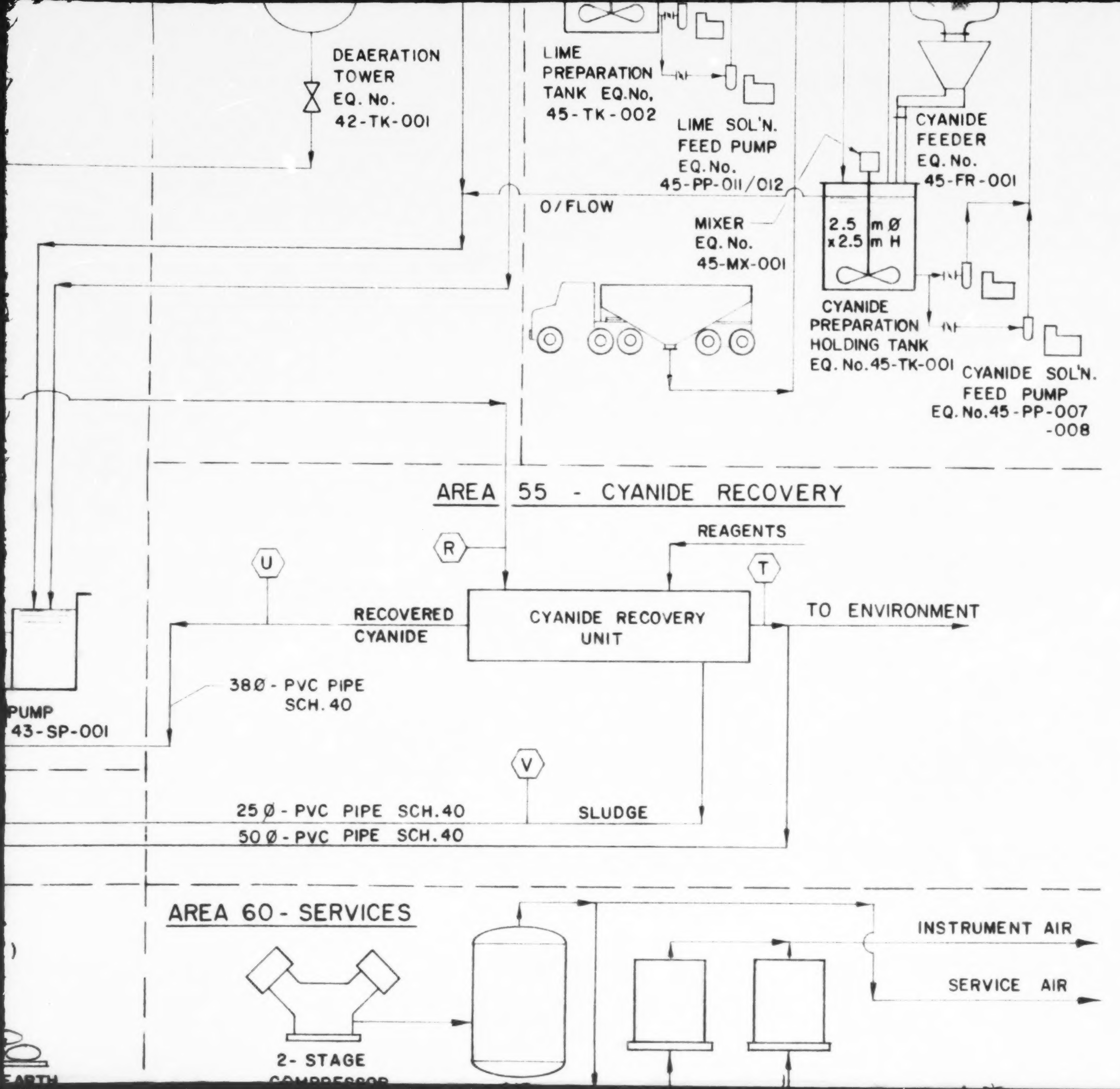
2. FLOW RATES AS SHOWN ARE FOR 743,200 TONNES HEAP.
3. PROCESS PLANT IS SCHEDULED TO RUN 24 HOURS PER DAY. 180 DAYS PER YEAR.
4. AREA DESIGNATIONS ARE LISTED AS FOLLOWS:

AREA 10	GENERAL
AREA 15	MINE / PIT
AREA 20	CRUSHING & SCREENING
AREA 30	LEACHING
AREA 40	PROCESS PLANT
AREA 41	CLARIFICATION
AREA 42	DEAERATION
AREA 43	ZINC PRECIPITATION
AREA 44	REAGENT STORAGE
AREA 45	REAGENT MIXING
AREA 46	LABORATORY
AREA 50	REFINERY
AREA 55	CYANIDE RECOVERY
AREA 60	SERVICES

5. FILTER PRECOAT (DIATOMACEOUS EARTH) TO BE ADDED AT START-UP AND AFTER FILTER CLOTH HAS BEEN CHANGED. PRECOAT CONSUMPTION WILL BE APPROX. 1000 kg/month (100 gramme / tonne)
6. Na CN CONSUMPTION ≅ 2.0 kg/tonne ORE.
7. LIME CONSUMPTION ≅ 4 kg/tonne ORE
8. REFINERY OPERATION: ONE - 8 HR FIRING EVERY WEEK.
9. TREATED EFFLUENT TO ENVIRONMENT CONTAINS < 1.0 ppm OF CYANIDE.


41-TK-004
TO BARREN SOL'N. POND
100-110-001
TO HEAP SURGE TANK
100-110-001
EQ. No. 30-TK-003
S
AREA 50 - REFINERY
VENT TO ATMOSPHERE
EXHAUST FAN
EQ. No. 50-EF-001
TO MINT
BULLION FURNACE
EQ. No. 50-BF-001
002
OIL BURNER
FUEL TANK
EQ. No. 50-PP-001
COMPRESSED AIR
SLAG JAW CRUSHER
EQ. No. 50-JC-001
FILTER PRESS
(1- SPARE)
EQ. No. 43- FP-001/002/003
ZINC BELT FEEDER
EQ. No. 43-FR-001
FE
FQ
P
PRECIPITATE FEED PUMP
(1 - SPARE)
EQ. No. 43-PP-001/002
Q
TO REAGENT MIX TANKS
O/FLOW
3 m Ø x
3 m H
BARREN SOLUTION TANK EQ. No. 43-TK-002
RECYCLE PUMP (1-SPARE)
EQ. No. 43-PP-003/004
SUMP PUMP
EQ. No. 43-S
TO BARREN SOL'N. POND
100-110-001
SLUDGE TO
100-110-001
SLUDGE IMPOUND POND
TO IMPOUND POND
100-110-001
AREA 44 - REAGENT STORAGE
(PROCESS BUILDING & STORAGE)


ZINC BELT FEEDER
EQ.No.43-FR-001
DEAERATION TOWER EQ. No. 42-TK-001
LIME PREPARATION TANK EQ.No. 45-TK-002
LIME SOL'N. FEED PUMP EQ.No. 45-PP-011/012
CYANIDE FEEDER EQ.No. 45-FR-001
FE
FQ
P
O/FLOW
MIXER EQ.No. 45-MX-001
2.5 m Ø x 2.5 m H
PRECIPITATE FEED PUMP (1 - SPARE) EQ.No. 43-PP-001/002
CYANIDE PREPARATION HOLDING TANK EQ.No.45-TK-001
CYANIDE SOL'N. FEED PUMP EQ.No.45-PP-007 -008
FILTER PRESS (1 - SPARE) EQ.No. 43-FP-001/002/003
Q
TO REAGENT MIX TANKS
O/FLOW
3 m Ø x 3 m H
BARREN SOLUTION TANK EQ.No. 43-TK-002
RECYCLE PUMP (1-SPARE) EQ.No. 43-PP-003/004
SUMP PUMP EQ.No. 43-SP-001
AREA 55 - CYANIDE RECOVERY
R
REAGENTS
U
T
RECOVERED CYANIDE
CYANIDE RECOVERY UNIT
TO ENVIRONMENT
38Ø - PVC PIPE SCH.40
V
25Ø - PVC PIPE SCH.40
SLUDGE
50Ø - PVC PIPE SCH.40
AREA 44 - REAGENT STORAGE
(PROCESS BUILDING & STORAGE)
CYANIDE (NaCN)
DIATOMACEOUS EARTH
AREA 60 - SERVICES
2- STAGE COMPRESSOR
INSTRUMENT AIR
SERVICE AIR
5.
6.
7.
8.
9.

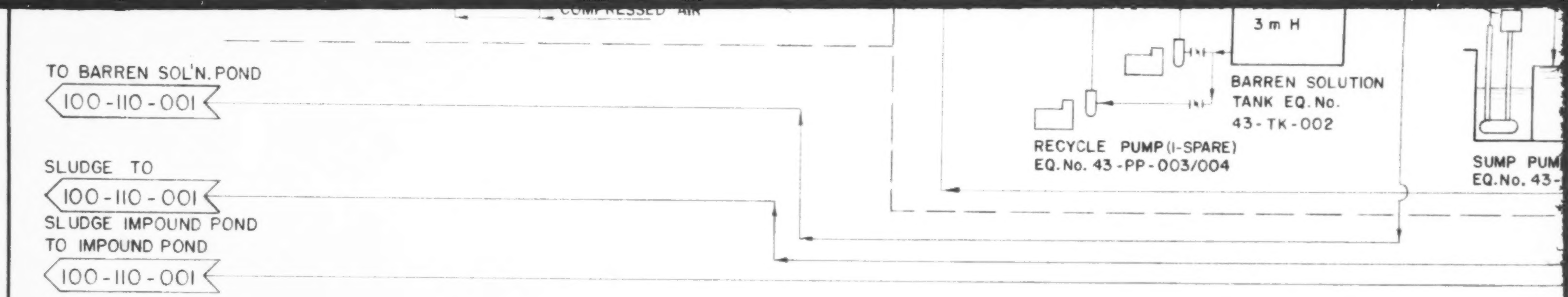

DEAERATION
TOWER
EQ. No.
42-TK-001
LIME
PREPARATION
TANK EQ.No.
45-TK-002
LIME SOL'N.
FEED PUMP
EQ.No.
45-PP-011/012
CYANIDE
FEEDER
EQ.No.
45-FR-001
O/FLOW
MIXER
EQ. No.
45-MX-001
2.5 m Ø
x 2.5 m H
CYANIDE
PREPARATION
HOLDING TANK
EQ. No. 45-TK-001
CYANIDE SOL'N.
FEED PUMP
EQ. No. 45-PP-007
-008
AREA 55 - CYANIDE RECOVERY
R
REAGENTS
U
T
RECOVERED
CYANIDE
CYANIDE RECOVERY
UNIT
TO ENVIRONMENT
380 - PVC PIPE
SCH. 40
PUMP
43-SP-001
V
25 Ø - PVC PIPE SCH. 40
SLUDGE
50 Ø - PVC PIPE SCH. 40
AREA 60 - SERVICES
INSTRUMENT AIR
SERVICE AIR
2- STAGE
COMPRESSOR
EARTH

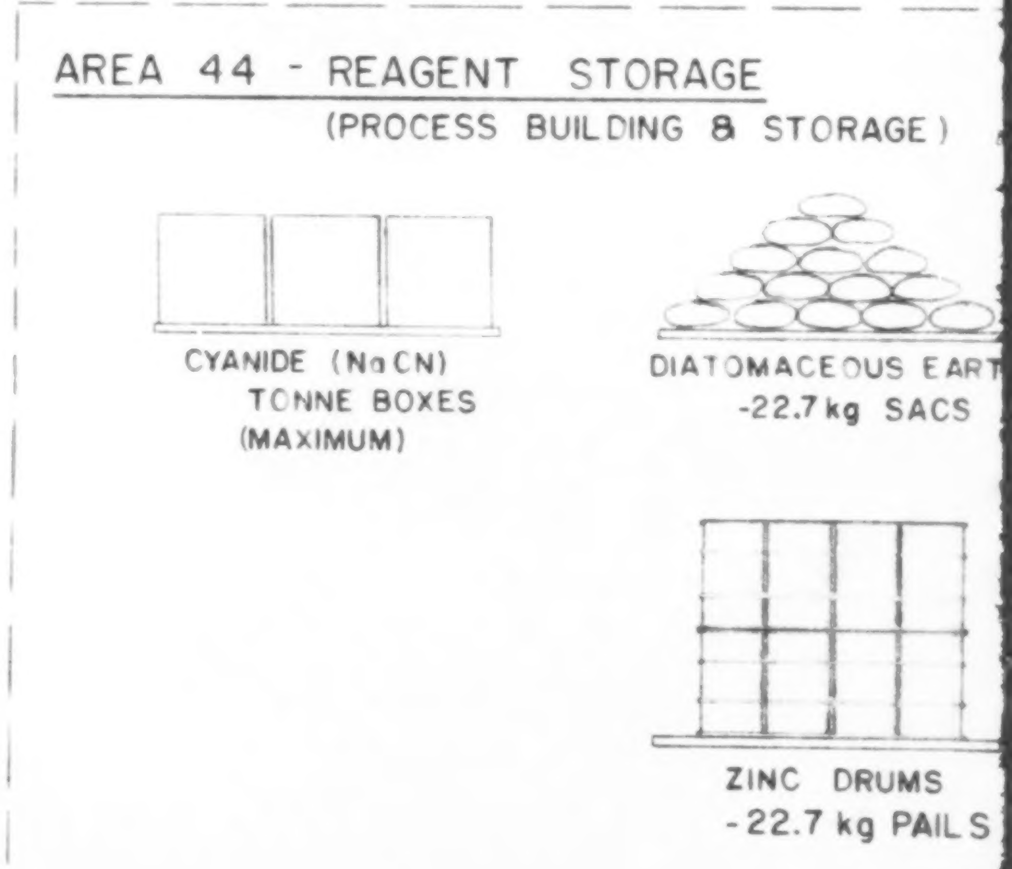
AREA 46	LABORATORY
AREA 50	REFINERY
AREA 55	CYANIDE RECOVERY
AREA 60	SERVICES

5. FILTER PRECOAT (DIATOMACEOUS EARTH) TO BE ADDED AT START-UP AND AFTER FILTER CLOTH HAS BEEN CHANGED. PRECOAT CONSUMPTION WILL BE APPROX. 1000 kg/month (100 gramme / tonne)
6. Na CN CONSUMPTION ≅ 2.0 kg/tonne ORE.
7. LIME CONSUMPTION ≅ 4 kg/tonne ORE
8. REFINERY OPERATION: ONE - 8 HR FIRING EVERY WEEK.
9. TREATED EFFLUENT TO ENVIRONMENT CONTAINS < 1.0 ppm OF CYANIDE.

LEGEND:

ITEM	DESCRIPTION

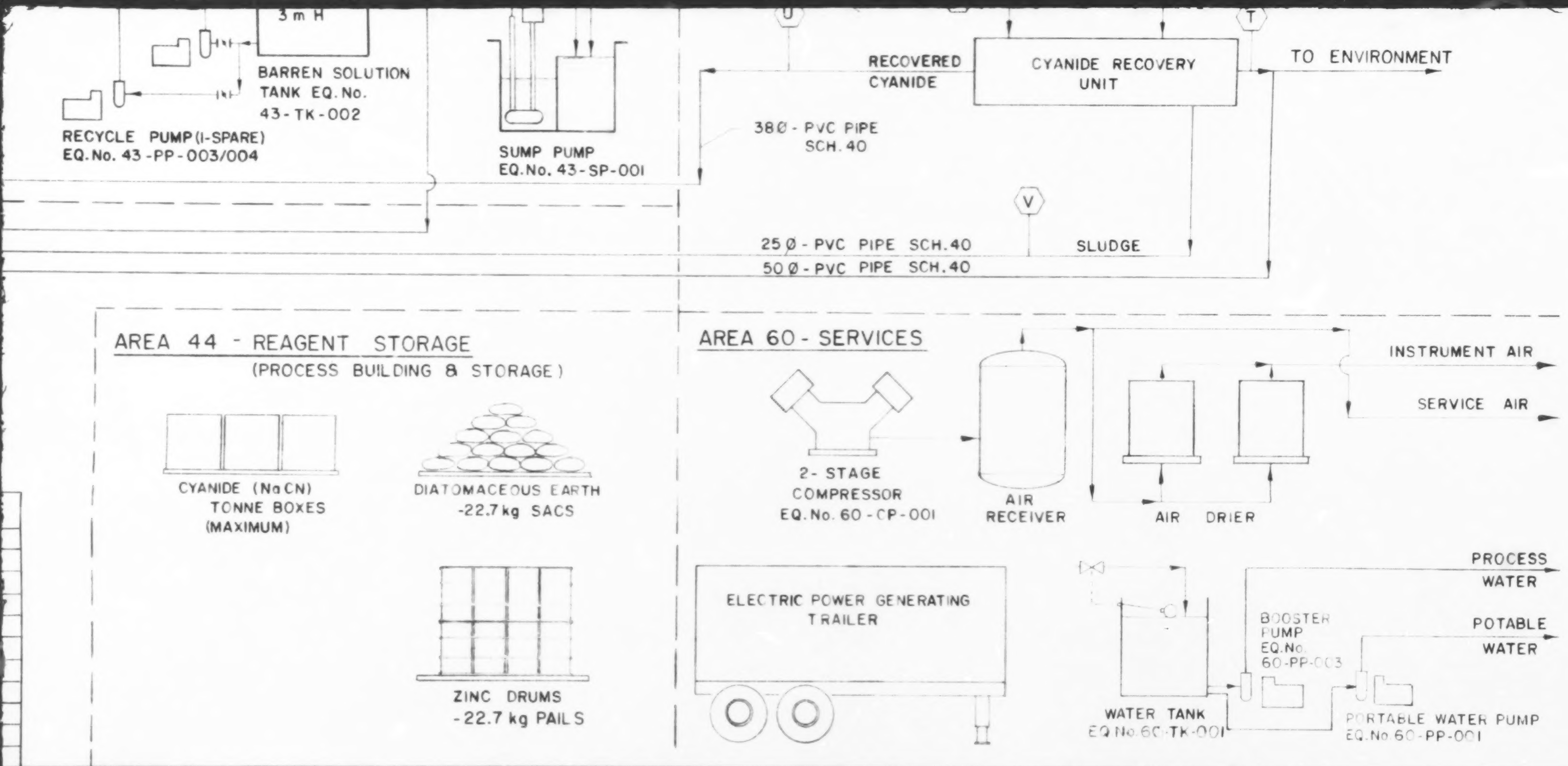

COMPRESSED AIR
3 m H
TO BARREN SOL'N. POND
100-110-001
BARREN SOLUTION
TANK EQ. No.
43-TK-002
RECYCLE PUMP (I-SPARE)
EQ. No. 43-PP-003/004
SUMP PUM
EQ.No. 43-
SLUDGE TO
100-110-001
SLUDGE IMPOUND POND
TO IMPOUND POND
100-110-001


AREA 44 - REAGENT STORAGE
(PROCESS BUILDING & STORAGE)
CYANIDE (NaCN)
TONNE BOXES
(MAXIMUM)
DIATOMACEOUS EART
-22.7 kg SACS
ZINC DRUMS
-22.7 kg PAILS

VARIABLES / STREAM	L	M	N	P	Q	R	S	T	U	V
DRY TONNES SOLIDS/HR	-	-	0.013	-	-	-	-	-	-	-
BULK SPECIFIC WEIGHT kg/m³	-		-	-	-	-	-	-	-	-
LIQUID FLOW liters/second	22.1	22.58	2	22.66	22.66	3.33	19.33	3.33	0.03	0.141
% SOLIDS BY WEIGHT	0	0	80	0.02	-	-	-	-	0	0.67
GOLD grams/min	3.81	4.14	3.01	3.81	0.007	0.001	0.006			
GOLD ppm	2.87	2.81	17.3	2.81	.005	.005	.005			
SPECIFIC GRAVITY	1.0		7.3	1.0	1.0	1.0	1.0	1.0	1.0	1.10
ph OF SOLUTION	11.0		11.0	11.0	11,0	11.0	11.0	10.0	11.0	10.0
TOTAL CYANIDE ppm	507						456	<1.0		<1.0

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHE
40-130-002	PROCESS PLANT LAYOUT - SECTION							
40-130-001	PROCESS PLANT LAYOUT - PLAN							
10-120-002	LOCAL SITE LAYOUT							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND							
30-130-001	HEAP GENERAL LAYOUT							
10-120-001	GENERAL SITE PLAN							
10-110-001	FLOW DIAGRAM (SHEET 1 OF 2)							

TRANSTEX


3 m H
BARREN SOLUTION
TANK EQ. No.
43-TK-002
RECYCLE PUMP (I-SPARE)
EQ. No. 43-PP-003/004
SUMP PUMP
EQ. No. 43-SP-001
T
RECOVERED
CYANIDE
CYANIDE RECOVERY
UNIT
TO ENVIRONMENT
380 - PVC PIPE
SCH. 40
V
25 0 - PVC PIPE SCH. 40
SLUDGE
50 0 - PVC PIPE SCH. 40
AREA 44 - REAGENT STORAGE
(PROCESS BUILDING & STORAGE)
CYANIDE (NaCN)
TONNE BOXES
(MAXIMUM)
DIATOMACEOUS EARTH
-22.7 kg SACS
ZINC DRUMS
-22.7 kg PAILS
AREA 60 - SERVICES
2- STAGE
COMPRESSOR
EQ. No. 60-CP-001
AIR
RECEIVER
AIR DRIER
INSTRUMENT AIR
SERVICE AIR
ELECTRIC POWER GENERATING
TRAILER
PROCESS
WATER
POTABLE
WATER
BOOSTER
PUMP
EQ. No.
60-PP-003
WATER TANK
EQ No. 60-TK-001
PORTABLE WATER PUMP
EQ. No. 60-PP-001

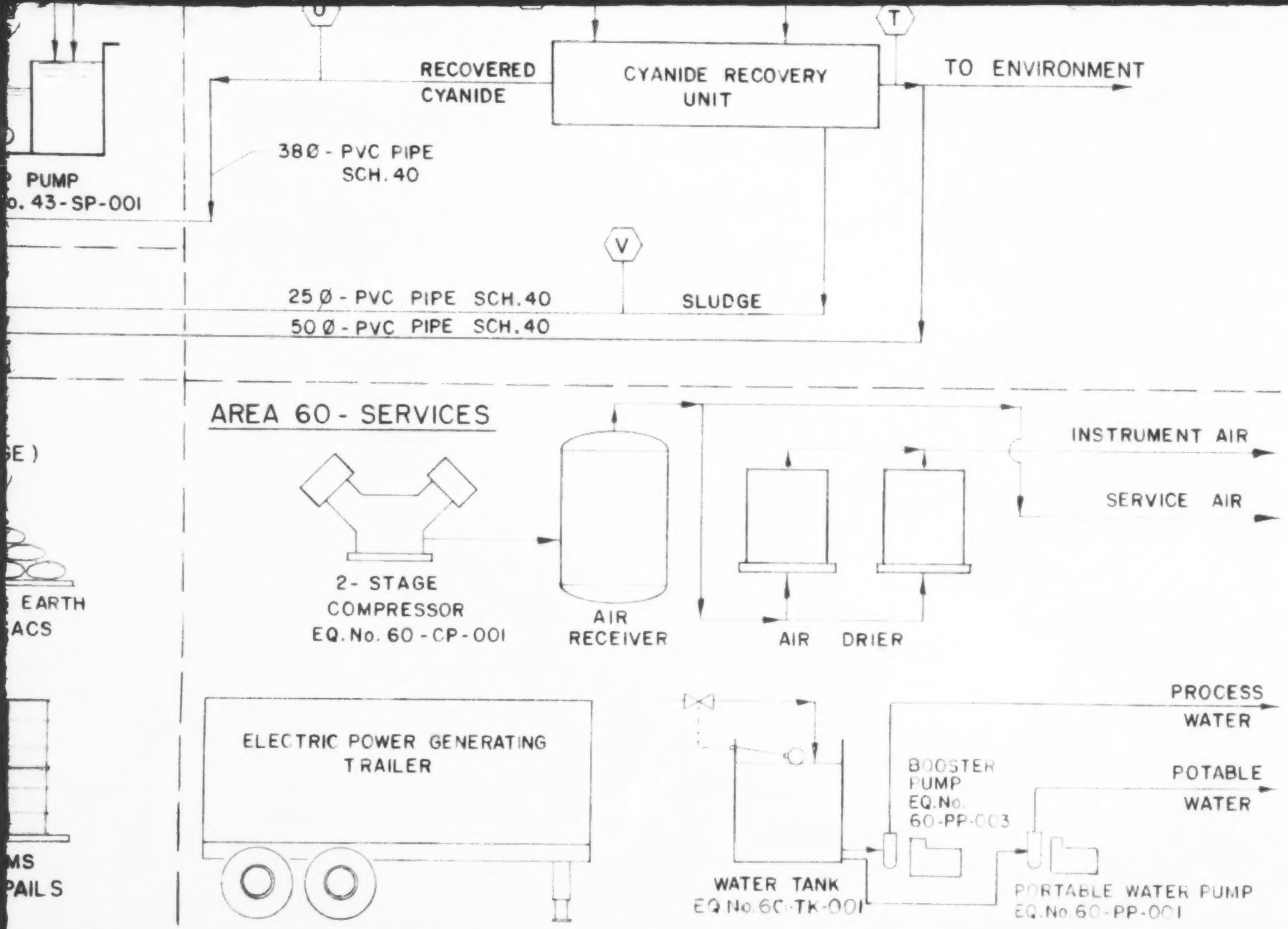
No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
					P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
					PI		RELEASED FOR INFORMATION	E.N.	

SCALE	NONE
DRAWN	E.N.
CHECKED	
APPROVAL	

DS

FL

DATE


T
V
RECOVERED CYANIDE
CYANIDE RECOVERY UNIT
TO ENVIRONMENT
380 - PVC PIPE SCH. 40
PUMP
o. 43-SP-001
25 0 - PVC PIPE SCH. 40
SLUDGE
50 0 - PVC PIPE SCH. 40
AREA 60 - SERVICES
E)
EARTH
ACS
MS
PAILS
2- STAGE COMPRESSOR
EQ. No. 60 - CP - 001
AIR RECEIVER
AIR DRIER
INSTRUMENT AIR
SERVICE AIR
ELECTRIC POWER GENERATING TRAILER
PROCESS WATER
POTABLE WATER
BOOSTER PUMP
EQ. No. 60-PP-003
WATER TANK
EQ No. 60-TK-001
PORTABLE WATER PUMP
EQ. No. 60-PP-001

LEGEND:

ITEM	DESCRIPTION
——	CONTINUOUS PROCESS FLOW
– – – –	FUTURE OR INTERMITTENT FLOW
⋈	GATE VALVE
⊣O⊢	BALL VALVE
⊣N⊢	CHECK VALVE
⊣/⊢	BUTTERFLY VALVE

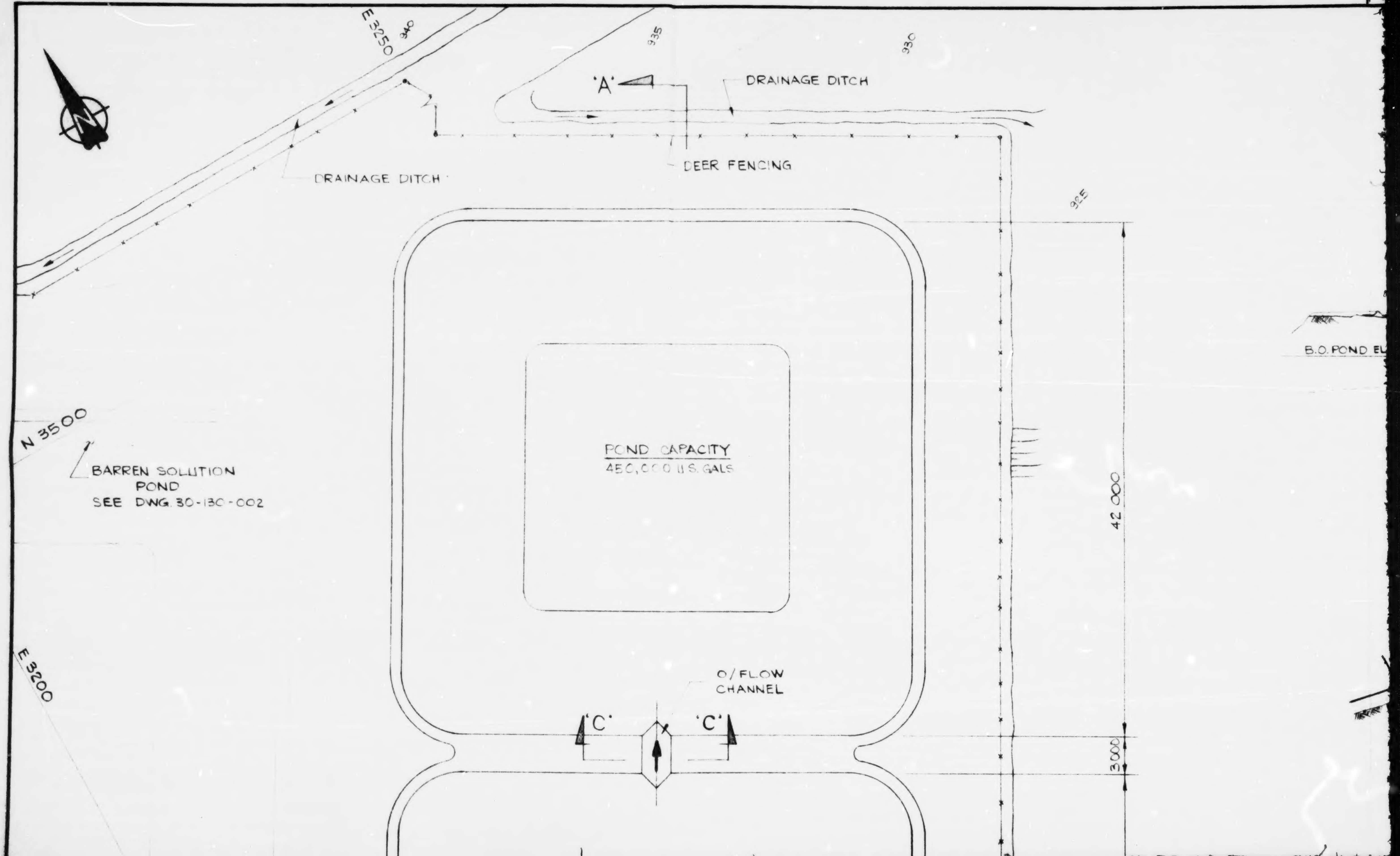
CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
	P1		RELEASED FOR INFORMATION	E.N.	

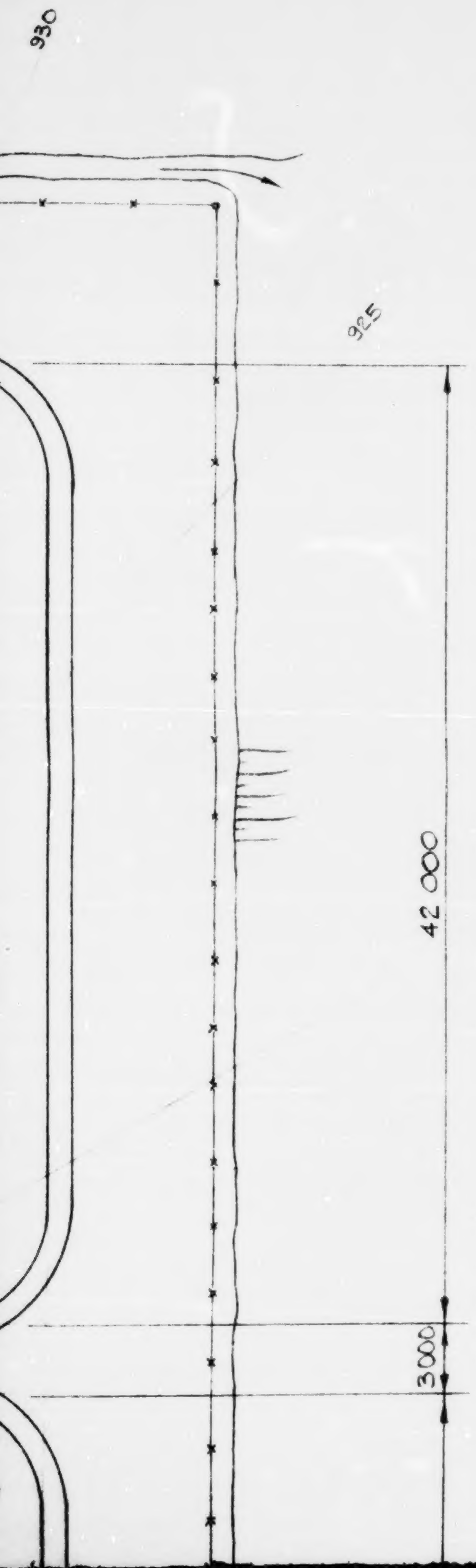
SCALE	NONE
DRAWN	E.N.
CHECKED	
APPROVAL	

WITTECK DEVELOPMENT INC.

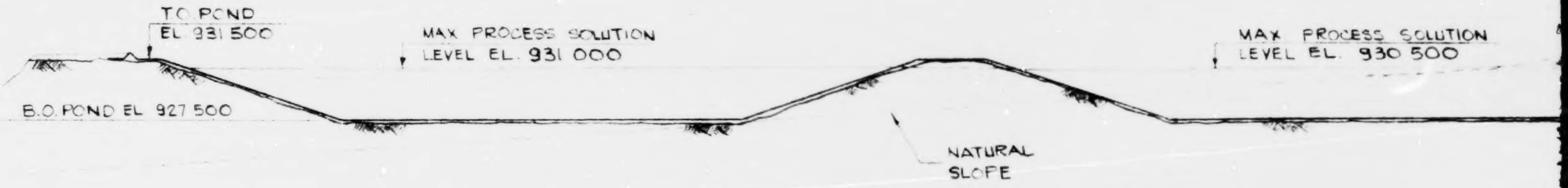
DSS-YUKON HEAP LEACH PROJECT
GENERAL (AREA No.10
HEAP LEACHING PROCESS
FLOW DIAGRAM (SHEET 2 OF 2)

DATE	PROJECT NO.	DWG. No.	REV.
	5209	10 - 110 - 002	P2


E 3250
940
935
930
'A'
DRAINAGE DITCH
DEER FENCING
DRAINAGE DITCH
925
B.O. POND EL
N 3500
BARREN SOLUTION POND
SEE DWG. 30-130-002
POND CAPACITY
450,000 U.S. GALS
42 000
O/FLOW CHANNEL
'C'
'C'
3000
E 3200

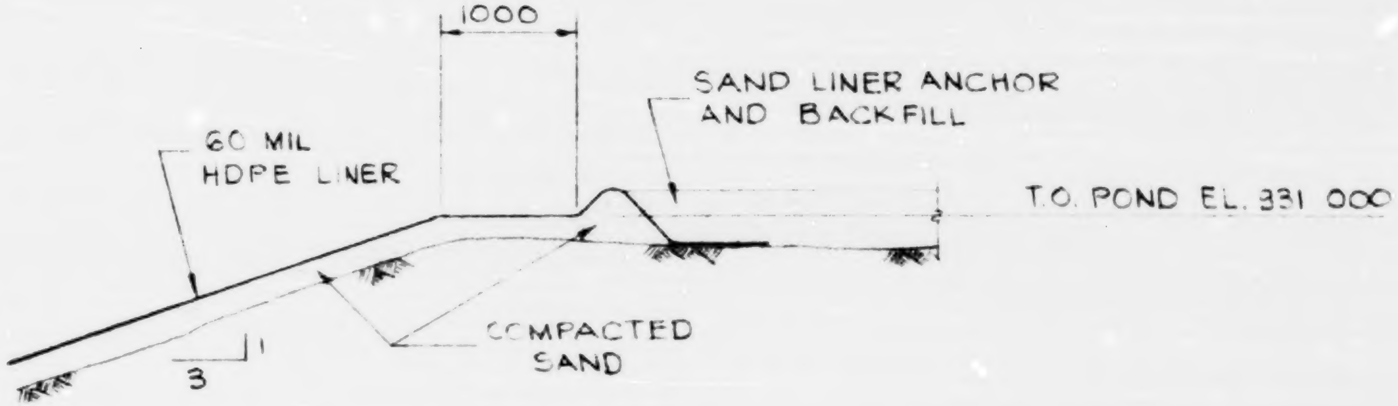


930
925
42 000
3000



T.O. POND
EL 331 500
MAX PROCESS SOLUTION
LEVEL EL. 931 000
B.O. POND EL 927 500
NATURAL
SLOPE
MAX PROCESS SOLUTION
LEVEL EL. 930 500

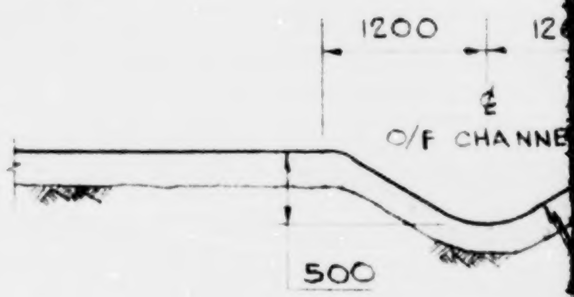
SECTION 'A - A'



1000
SAND LINER ANCHOR
AND BACKFILL
60 MIL
HDPE LINER
T.O. POND EL. 331 000
COMPACTED
SAND
3
1

DETAIL 'B'

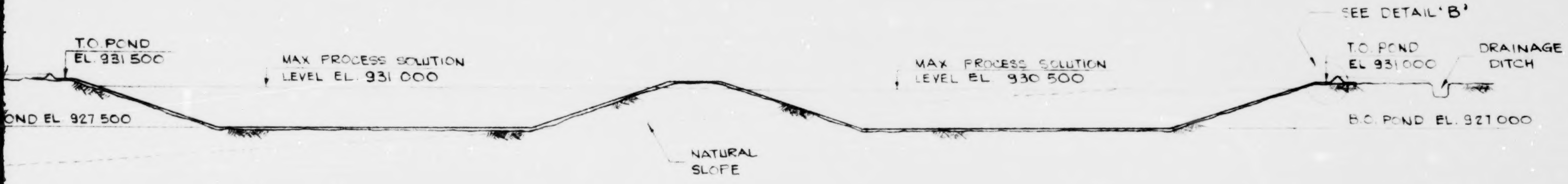
TYPICAL ANCHOR DETAIL
SCALE: 1:50



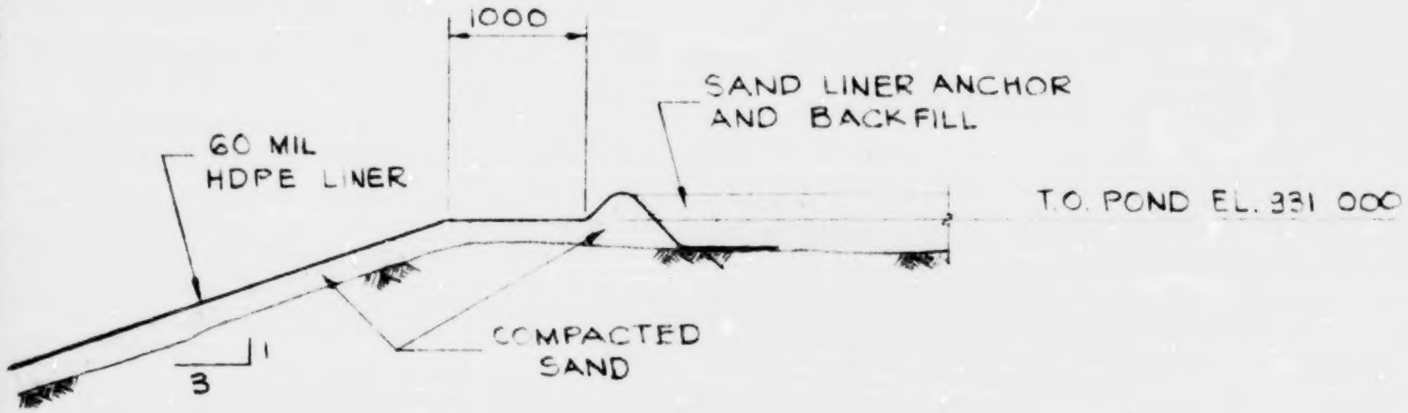
1200
12
℄
O/F CHANNE
500

SECTION

SCALE 1:


T.O. POND
EL. 931 500
MAX PROCESS SOLUTION
LEVEL EL. 931 000
OND EL. 927 500
NATURAL
SLOPE
MAX PROCESS SOLUTION
LEVEL EL. 930 500
SEE DETAIL 'B'
T.O. POND
EL. 931 000
DRAINAGE
DITCH
B.O. POND EL. 921 000

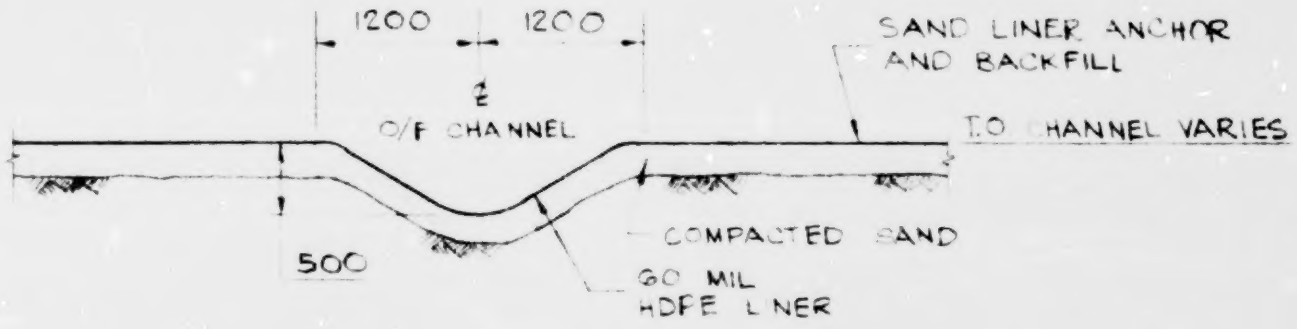
SECTION 'A - A'


1000
60 MIL
HDPE LINER
SAND LINER ANCHOR
AND BACKFILL
T.O. POND EL. 331 000
COMPACTED
SAND
1
3

DETAIL 'B'

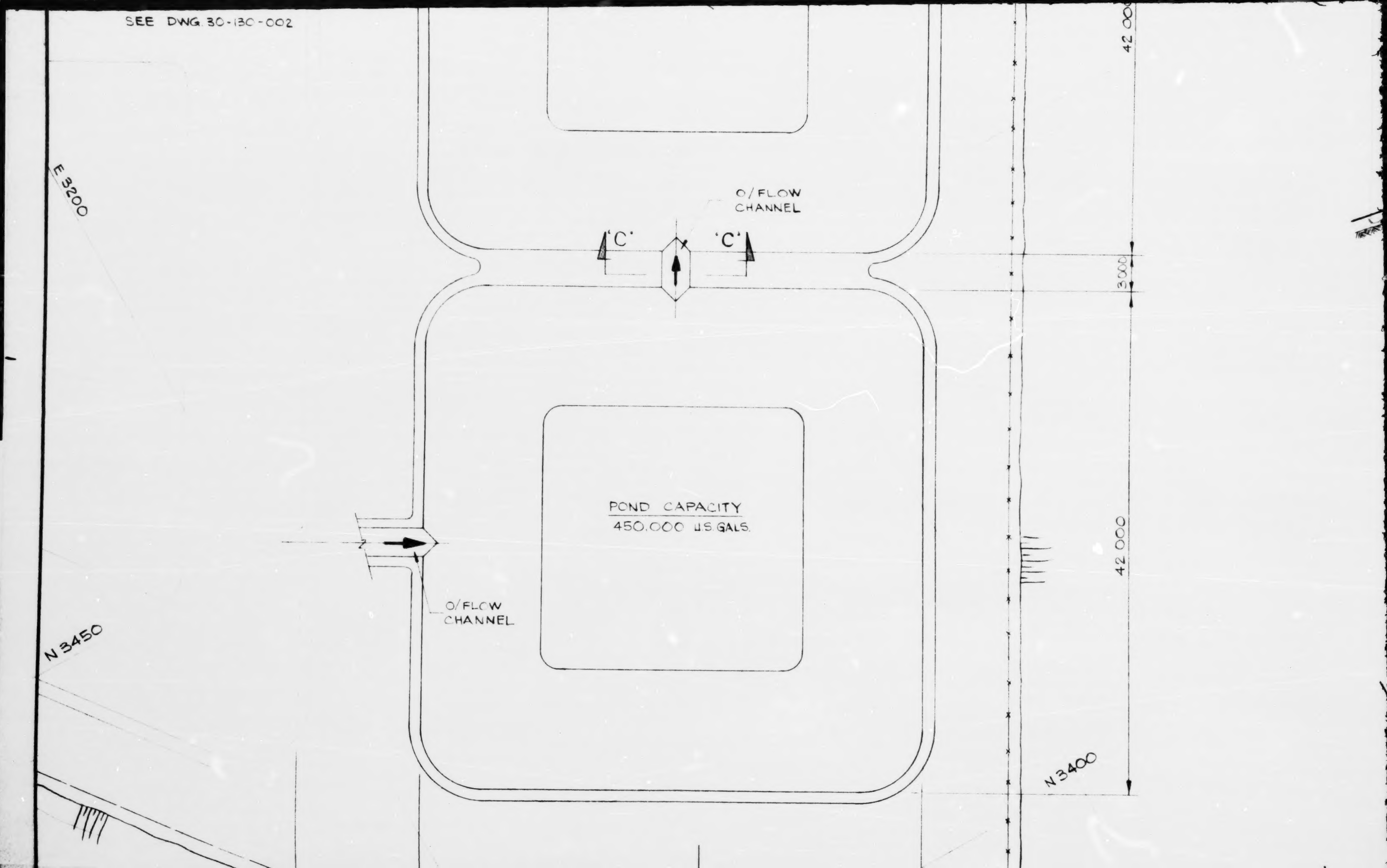
TYPICAL ANCHOR DETAIL

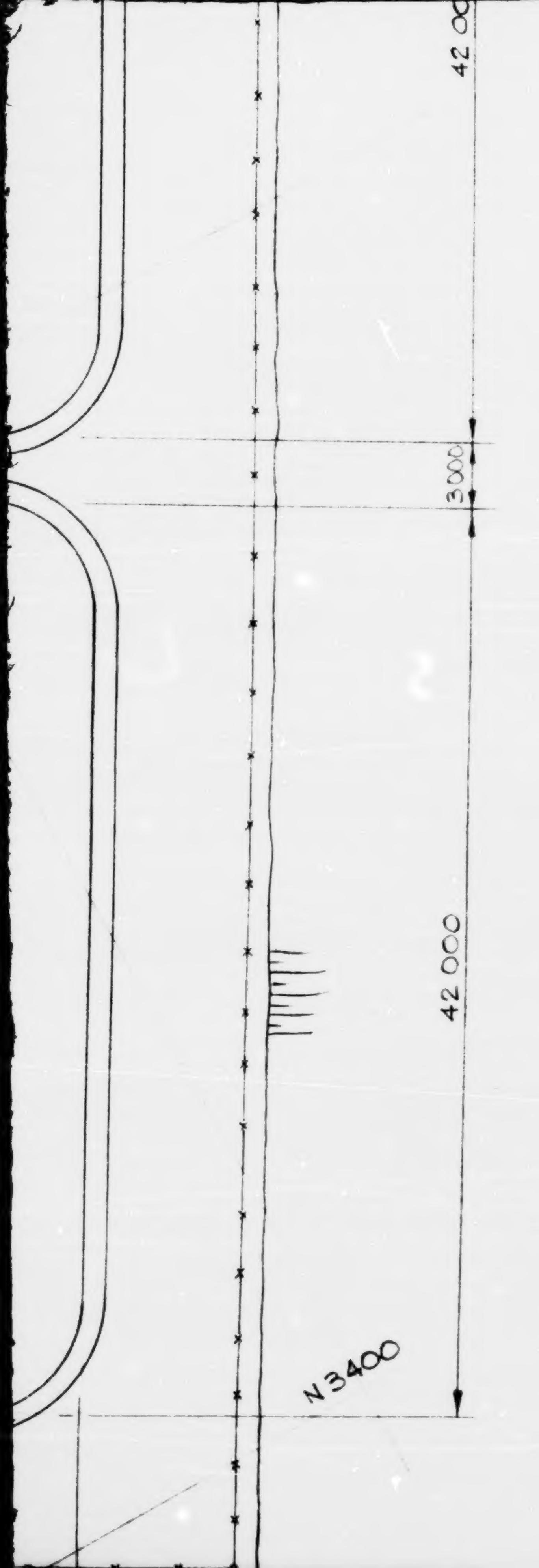
SCALE: 1:50

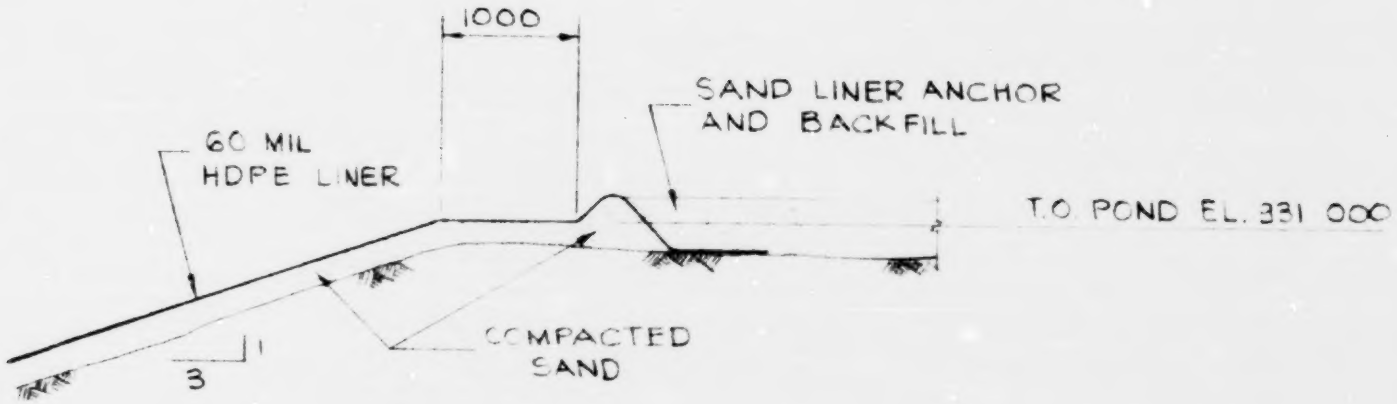

1200
1200
℄
O/F CHANNEL
SAND LINER ANCHOR
AND BACKFILL
T.O. CHANNEL VARIES
500
COMPACTED SAND
60 MIL
HDPE LINER

SECTION 'C-C'

SCALE 1:50


SEE DWG. 30-130-002
E 3200
O/FLOW
CHANNEL
'C'
'C'
42 000
3000
POND CAPACITY
450,000 U.S. GALS.
42 000
O/FLOW
CHANNEL
N 3450
N 3400

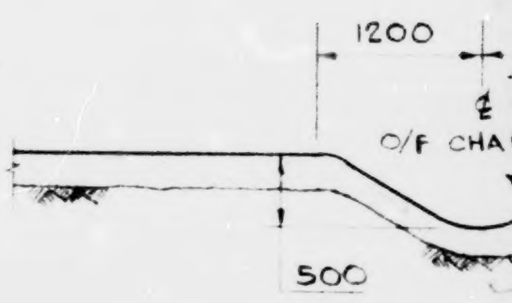

42 000
3000
42 000
N 3400


1000
60 MIL
HDPE LINER
SAND LINER ANCHOR
AND BACKFILL
T.O. POND EL. 331.000
COMPACTED
SAND
3
1

DETAIL 'B'

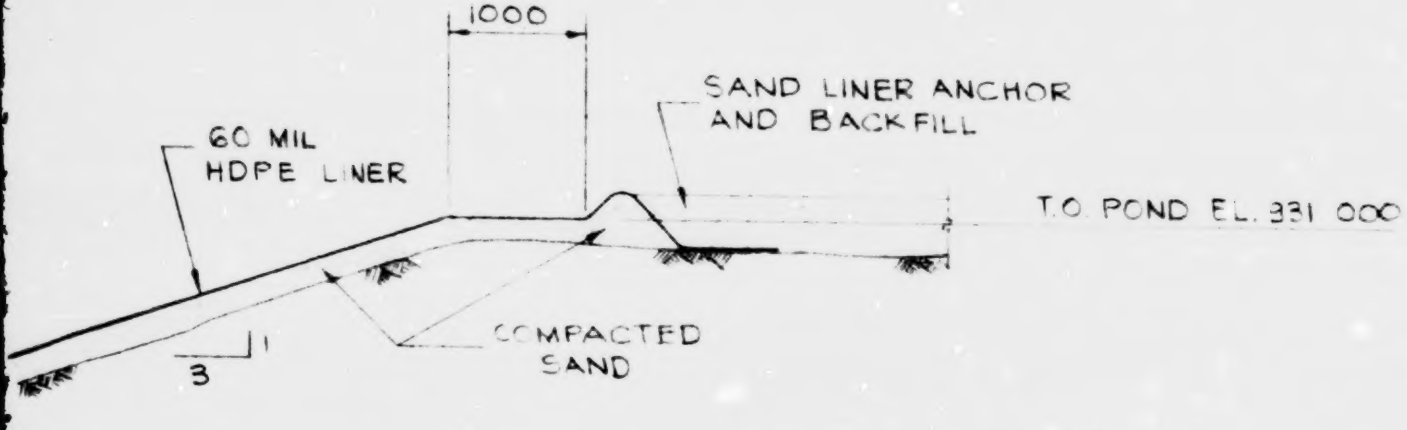
TYPICAL ANCHOR DETAIL

SCA[illegible] 1:50

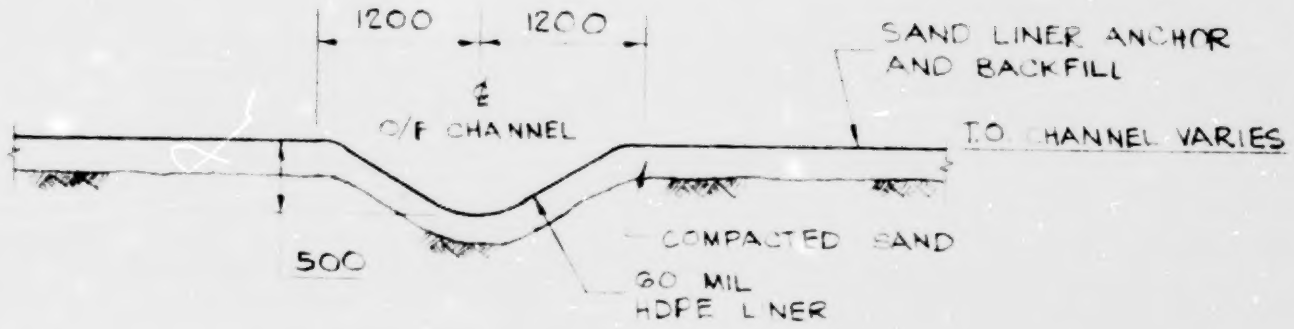

1200
℄
O/F CHA
500

SECTIO

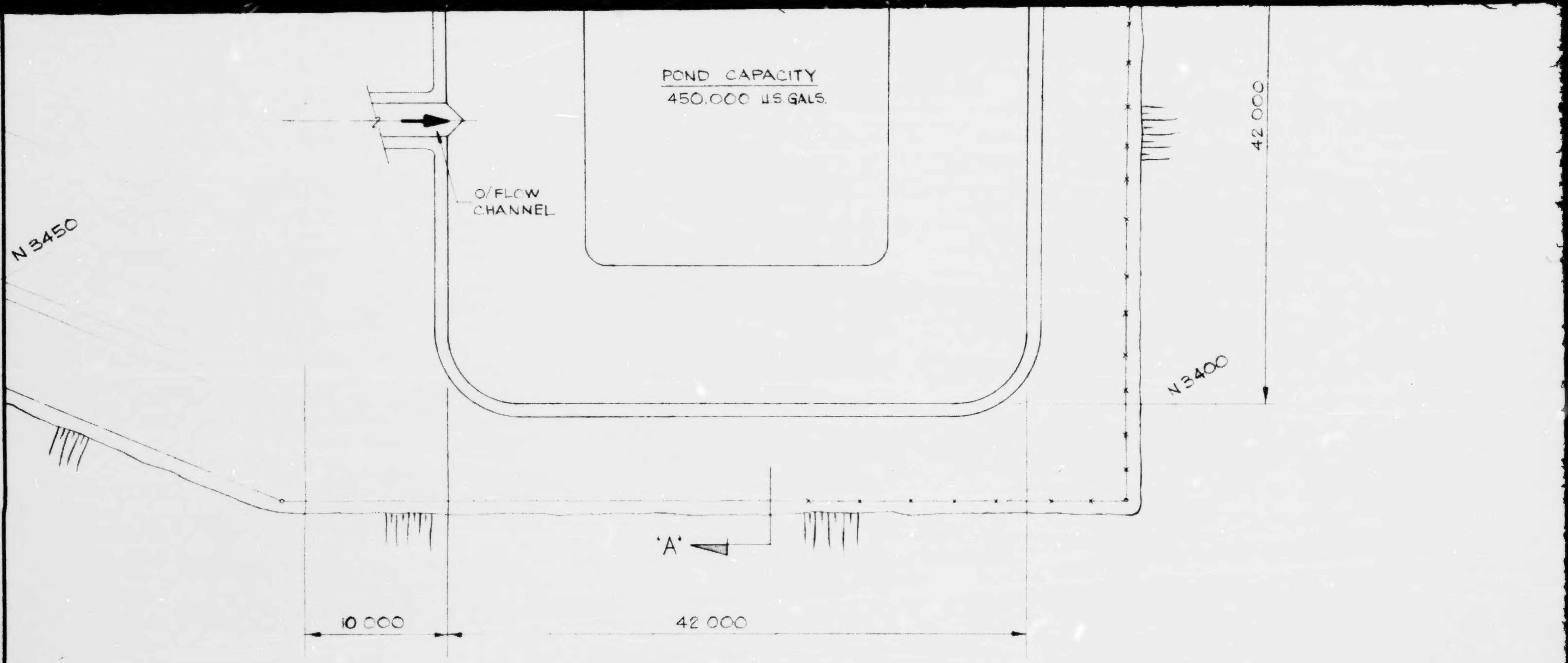
SCALE


1000
SAND LINER ANCHOR AND BACKFILL
60 MIL HDPE LINER
T.O. POND EL. 331 000
3
1
COMPACTED SAND

DETAIL 'B'
TYPICAL ANCHOR DETAIL
SCALE 1:50


1200
1200
℄ O/F CHANNEL
SAND LINER ANCHOR AND BACKFILL
T.O. CHANNEL VARIES
500
COMPACTED SAND
60 MIL HDPE LINER

SECTION 'C-C'
SCALE 1:50

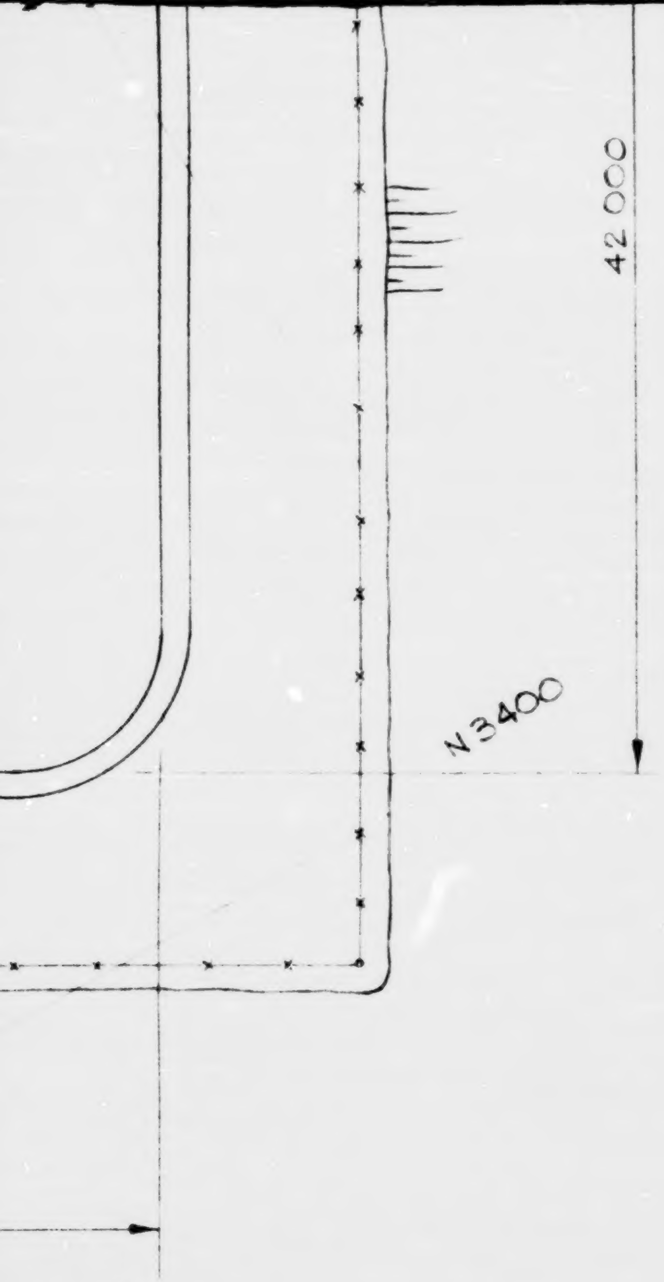

POND CAPACITY
450,000 U.S GALS.
O/FLOW
CHANNEL
N 3450
N 3400
42 000
10 000
42 000
'A'

IMPOUND POND

PLAN

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHE
40-130-002	PROCESS PLANT GENERAL LAYOUT - SECTION							
40-130-001	PROCESS PLANT GENERAL LAYOUT - PLAN							
30-130-004	LOCAL SITE LAYOUT							
30-130-002	BARREN SOLUTION POND GENERAL LAYOUT							
30-130-001	HEAP GENERAL LAYOUT							
10-120-001	GENERAL SITE LAYOUT							
10 -110-002	FLOW DIAGRAM (SHEET 2 OF 2)							
10 -110-001	FLOW DIAGRAM (SHEET 1 OF 2)	40						

TRANSTEX



	No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
						P2	MAY20,87	RELEASED FOR REPORT	J. LI	
						P1		RELEASED FOR INFORMATION	E.N.	

SCALE	1:250 U/N	
DRAWN	E.N.	JAN.5,1987
CHECKED		
APPROVAL		

DATE

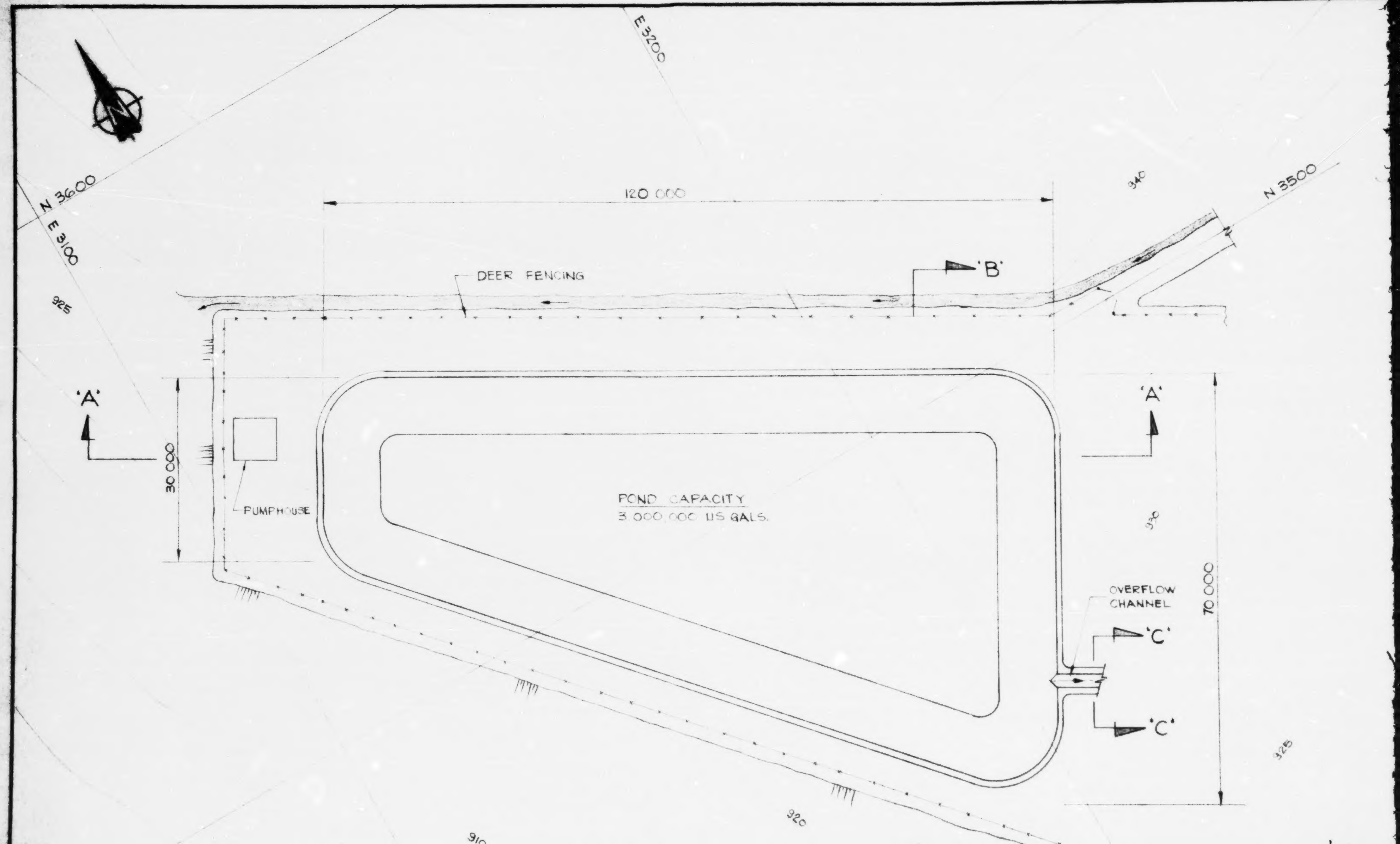
Y	CHECK	No.	DATE	REVISIONS	BY	CHECK
		P2	MAY20,87	RELEASED FOR REPORT	J. LI	
		P1		**RELEASED FOR INFORMATION**	E.N.	

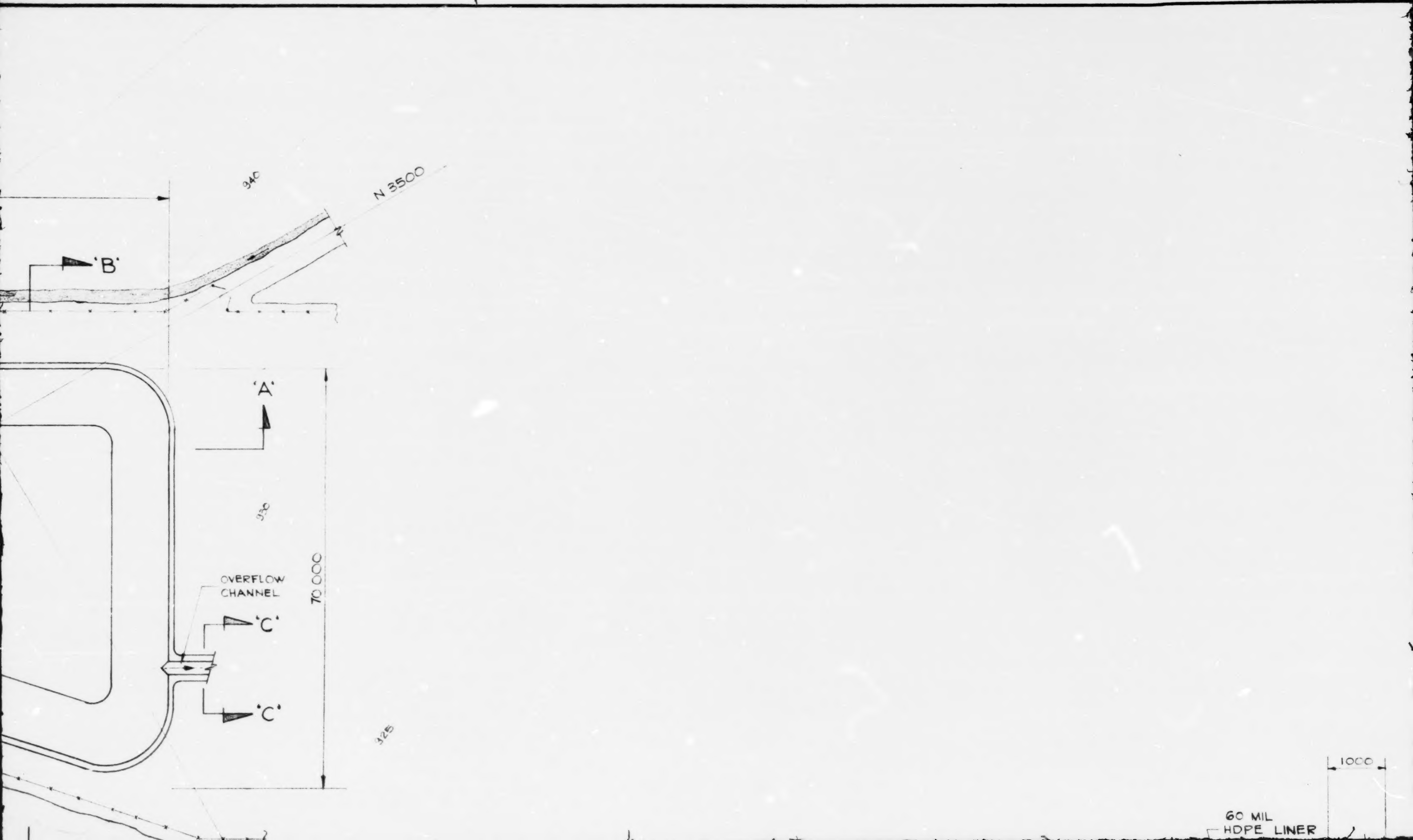
SCALE	1:250 U/N	
DRAWN	E.N.	JAN.5,1987
CHECKED		
APPROVAL		

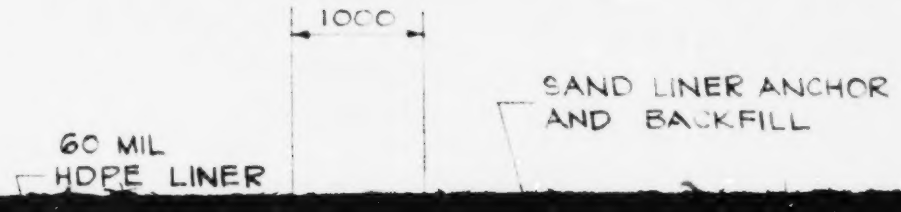
WITTECK DEVELOPMENT INC.

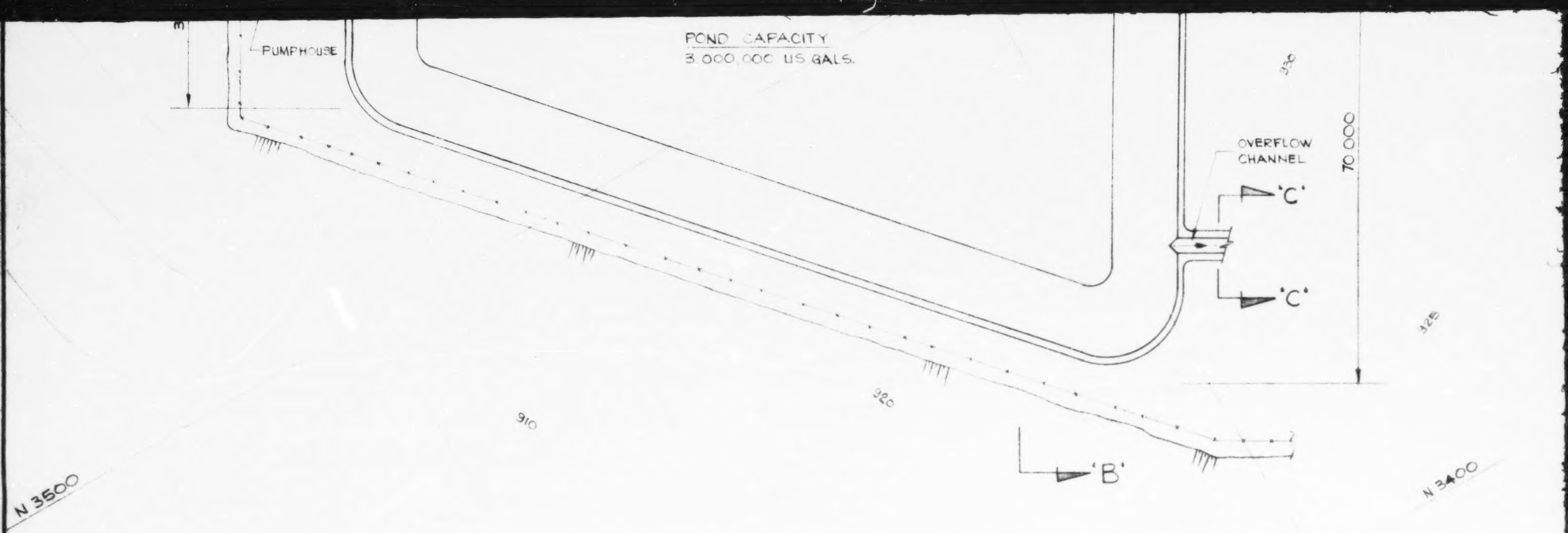
DSS - YUKON HEAP LEACH PROJECT
LEACHING (AREA 30)
IMPOUND POND GENERAL LAYOUT
PLAN AND SECTIONS

DATE	PROJECT NO.	DWG. No.	REV.
	5209	30 - 130 - 003	P2


E 3200
N 3600
E 3100
N 3500
940
925
120 000
DEER FENCING
'B'
'A'
'A'
30 000
PUMPHOUSE
POND CAPACITY
3 000,000 US GALS.
930
70 000
OVERFLOW CHANNEL
'C'
'C'
925
920
910

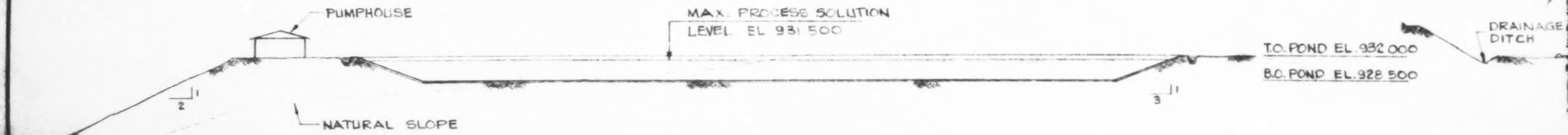

940
N 3500
'B'
'A'
930
OVERFLOW
CHANNEL
70 000
'C'
'C'
925
1000
60 MIL
HDPE LINER

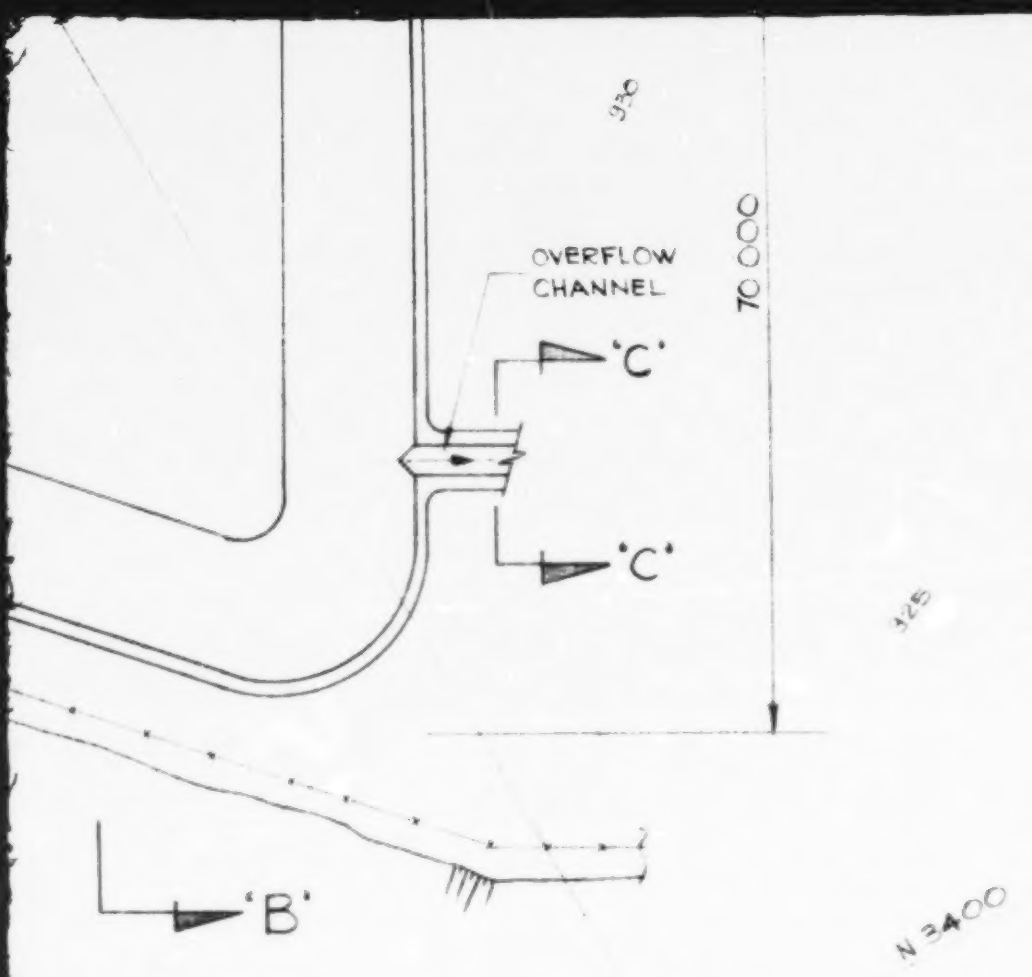

1000
SAND LINER ANCHOR
AND BACKFILL
60 MIL
HDPE LINER

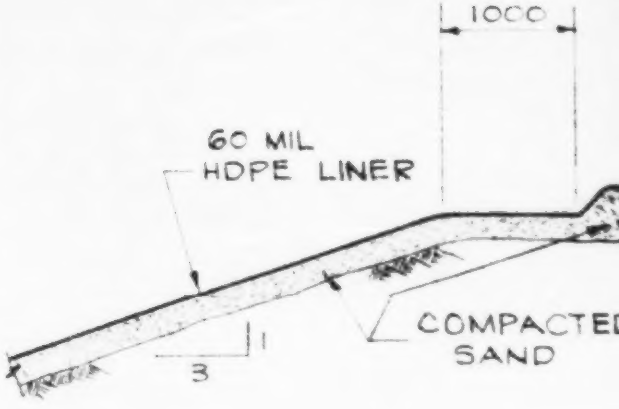

PUMPHOUSE
3
POND CAPACITY
3,000,000 US GALS.
930
70 000
OVERFLOW
CHANNEL
'C'
'C'
925
920
910
'B'
N 3500
N 3400

BARREN SOLUTION POND

PLAN


PUMPHOUSE
MAX. PROCESS SOLUTION
LEVEL EL 931.500
DRAINAGE
DITCH
T.O. POND EL. 932.000
B.O. POND EL. 928.500
2
1
3
1
NATURAL SLOPE

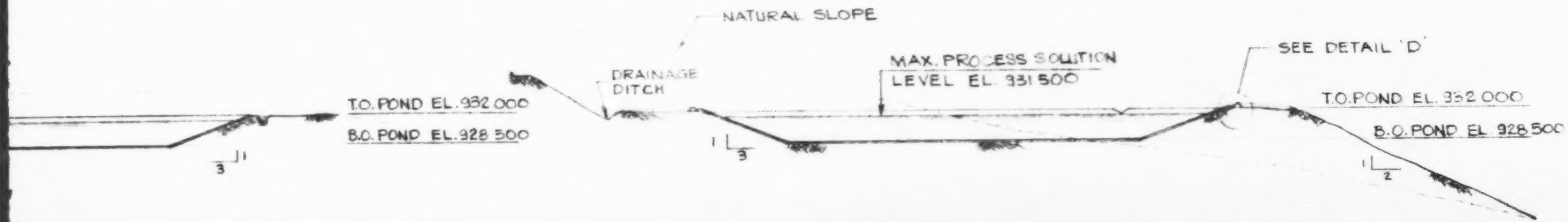

930
OVERFLOW
CHANNEL
70 000
'C'
'C'
325
'B'
N 3400

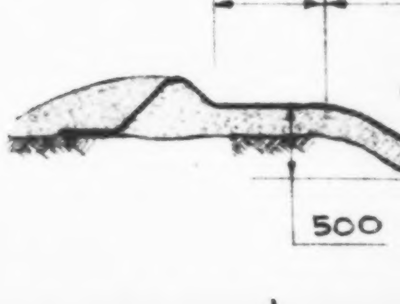

1000
60 MIL
HDPE LINER
1
3
COMPACTED
SAND

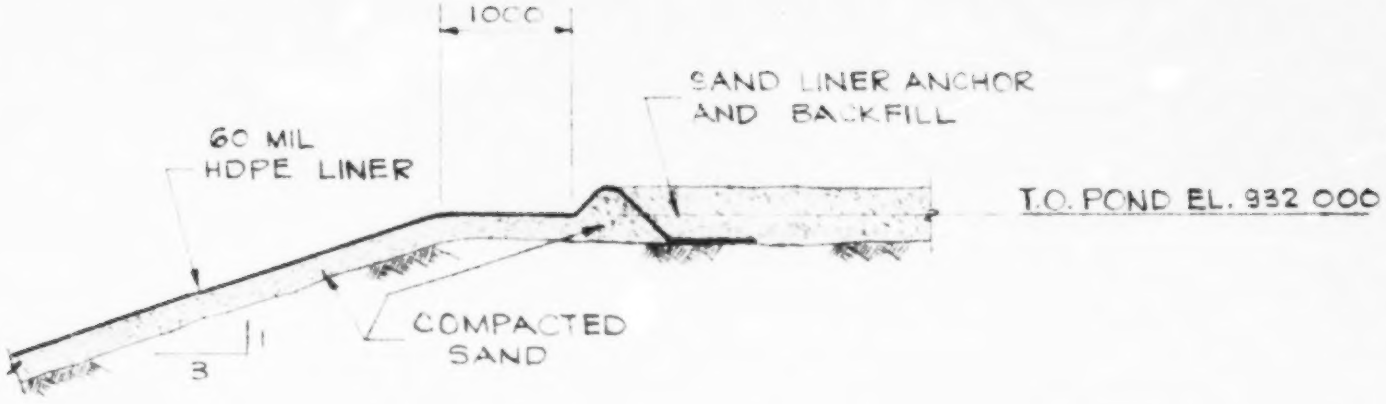
DETAI

TYPICAL AN

SCALE: I


NATURAL SLOPE
DRAINAGE
DITCH
MAX. PROCESS SOLUTION
LEVEL EL. 931 500
SEE DETAIL 'D'
T.O. POND EL. 932 000
B.O. POND EL. 928 500
3
1
1
3
T.O. POND EL. 932 000
B.O. POND EL. 928 500
1
2

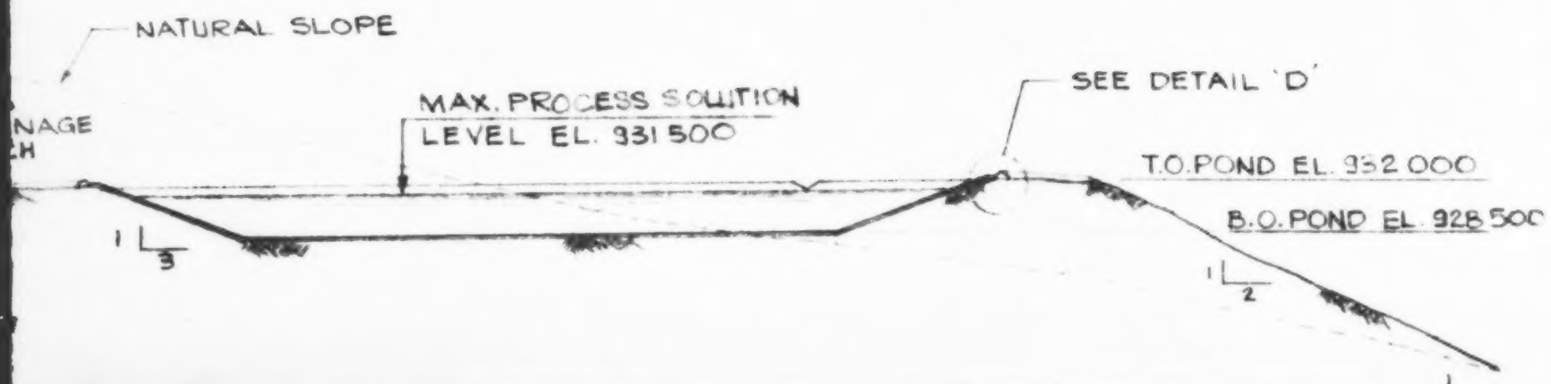


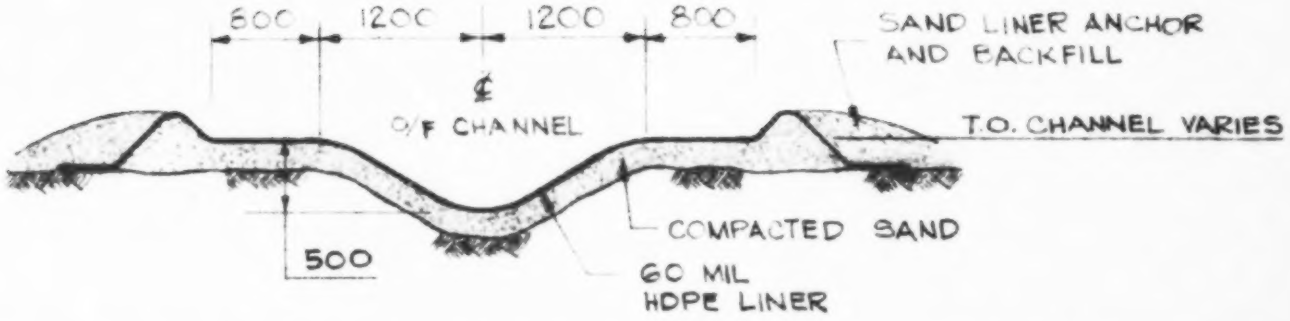

1000
SAND LINER ANCHOR AND BACKFILL
60 MIL HDPE LINER
T.O. POND EL. 932 000
COMPACTED SAND
1
3

DETAIL 'D'

TYPICAL ANCHOR DETAIL

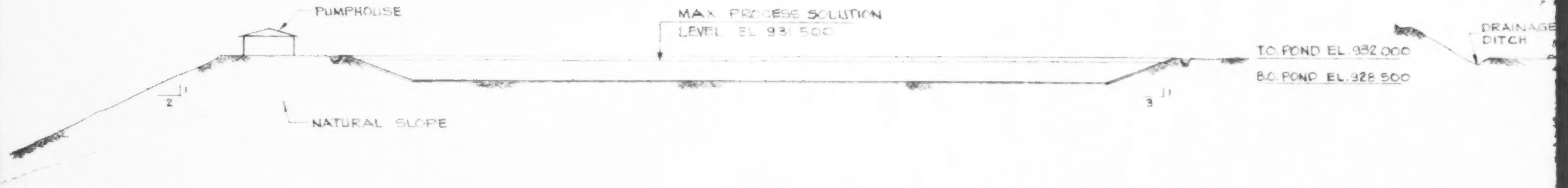
SCALE: 1:50


NATURAL SLOPE
NAGE
CH
MAX. PROCESS SOLUTION LEVEL EL. 931 500
SEE DETAIL 'D'
T.O. POND EL. 932 000
B.O. POND EL. 928 500
1
3
1
2


800
1200
1200
800
SAND LINER ANCHOR AND BACKFILL
℄ O/F CHANNEL
T.O. CHANNEL VARIES
COMPACTED SAND
500
60 MIL HDPE LINER

BARREN SOLUTION POND

PLAN


PUMPHOUSE
MAX. PROCESS SOLUTION LEVEL EL 931.500
T.O. POND EL 932.000
B.O. POND EL 928.500
DRAINAGE DITCH
NATURAL SLOPE
2
1
3
1

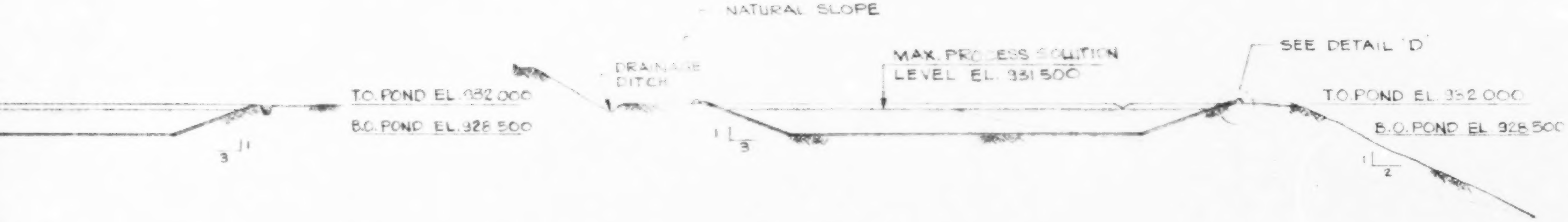
SECTION 'A-A'

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHE
40-130-002	PROCESS PLANT LAYOUT – SECTION							
40-130-001	PROCESS PLANT LAYOUT – PLAN							
10-120-002	LOCAL SITE LAYOUT							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-001	HEAP GENERAL LAYOUT							
10-120-001	GENERAL SITE PLAN							
10-110-002	FLOW DIAGRAM (SHEET 2 OF 2)							
10-110-001	FLOW DIAGRAM (SHEET 1 OF 2)							

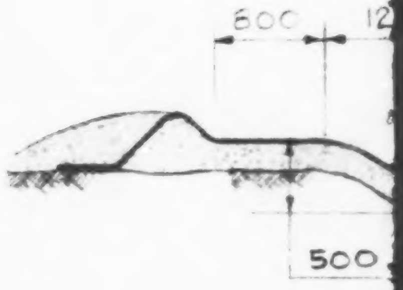
DETA

TYPICAL A

SCALE


NATURAL SLOPE
DRAINAGE DITCH
MAX. PROCESS SOLUTION LEVEL EL. 931.500
SEE DETAIL 'D'
T.O. POND EL. 932.000
B.O. POND EL. 928.500
T.O. POND EL. 932.000
B.O. POND EL. 928.500
3
1
1
3
1
2

SECTION 'B - B'



SE

	No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
						P2		RELEASED FOR REPORT	J. LI	
						P1		RELEASED FOR INFORMATION	E.N.	

SCALE	1:500 U/N	
DRAWN	E.NATIVIDAD	DEC. 19, 1986
CHECKED		
APPROVAL		

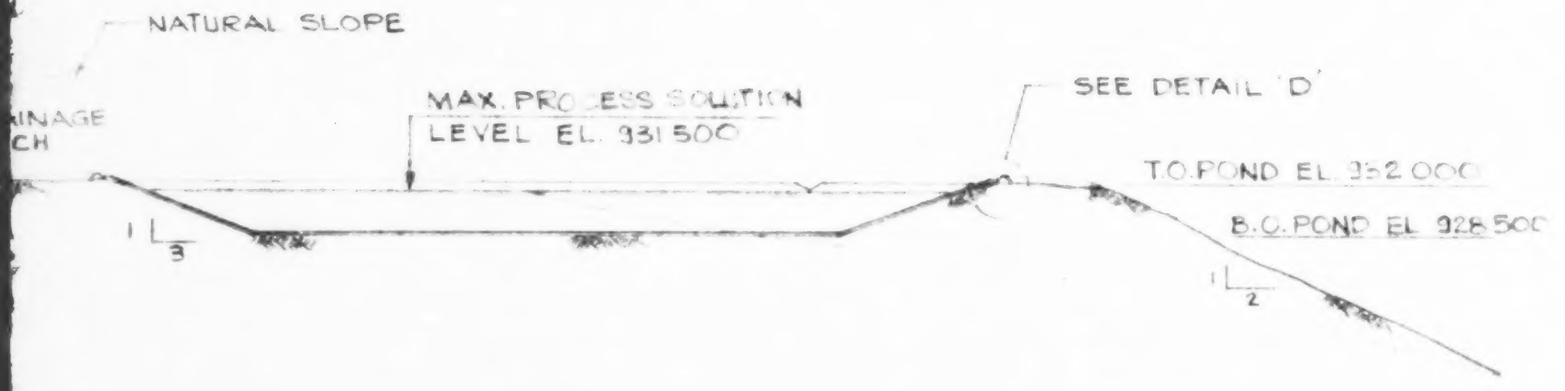
DS

BA

DATE

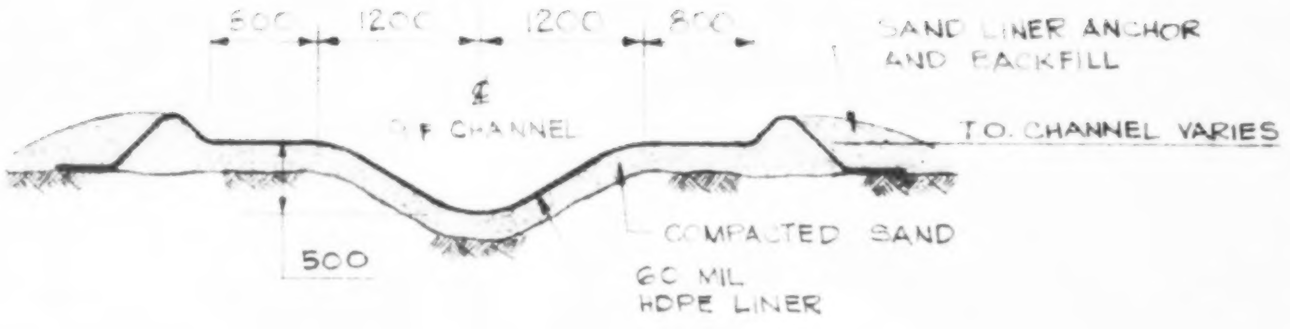
DETAIL 'D'

TYPICAL ANCHOR DETAIL
SCALE: 1:50



NATURAL SLOPE
INAGE
CH
MAX. PROCESS SOLUTION
LEVEL EL. 931.500
SEE DETAIL 'D'
T.O.POND EL. 932.000
B.O.POND EL. 928.500
1
3
1
2

SECTION 'B - B'



800
1200
1200
800
℄
OF CHANNEL
SAND LINER ANCHOR
AND BACKFILL
T.O. CHANNEL VARIES
500
COMPACTED SAND
60 MIL
HDPE LINER

SECTION 'C - C'
SCALE: 1:50

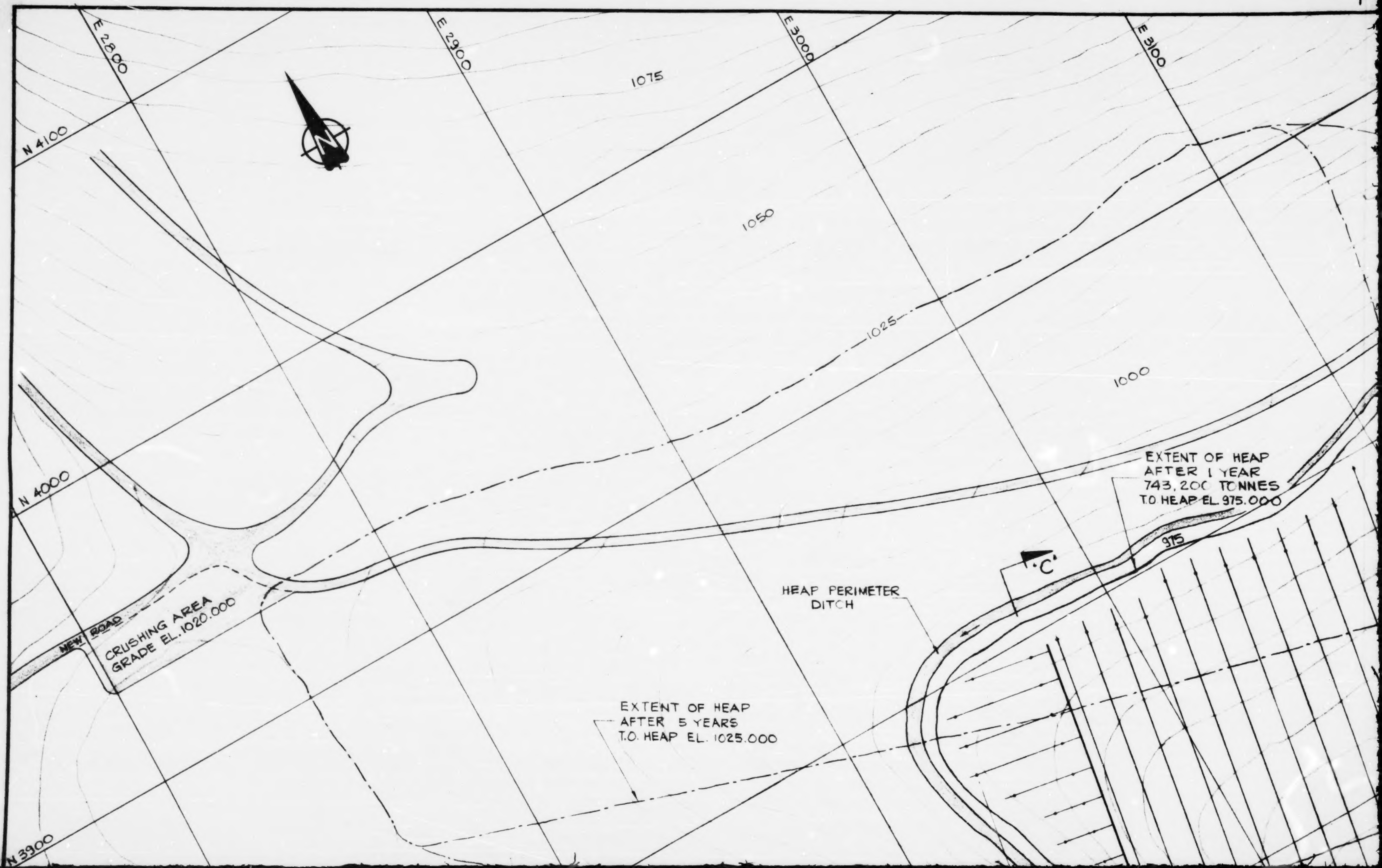
CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2		RELEASED FOR REPORT	J.LI	
	P1		RELEASED FOR INFORMATION	E.N.	

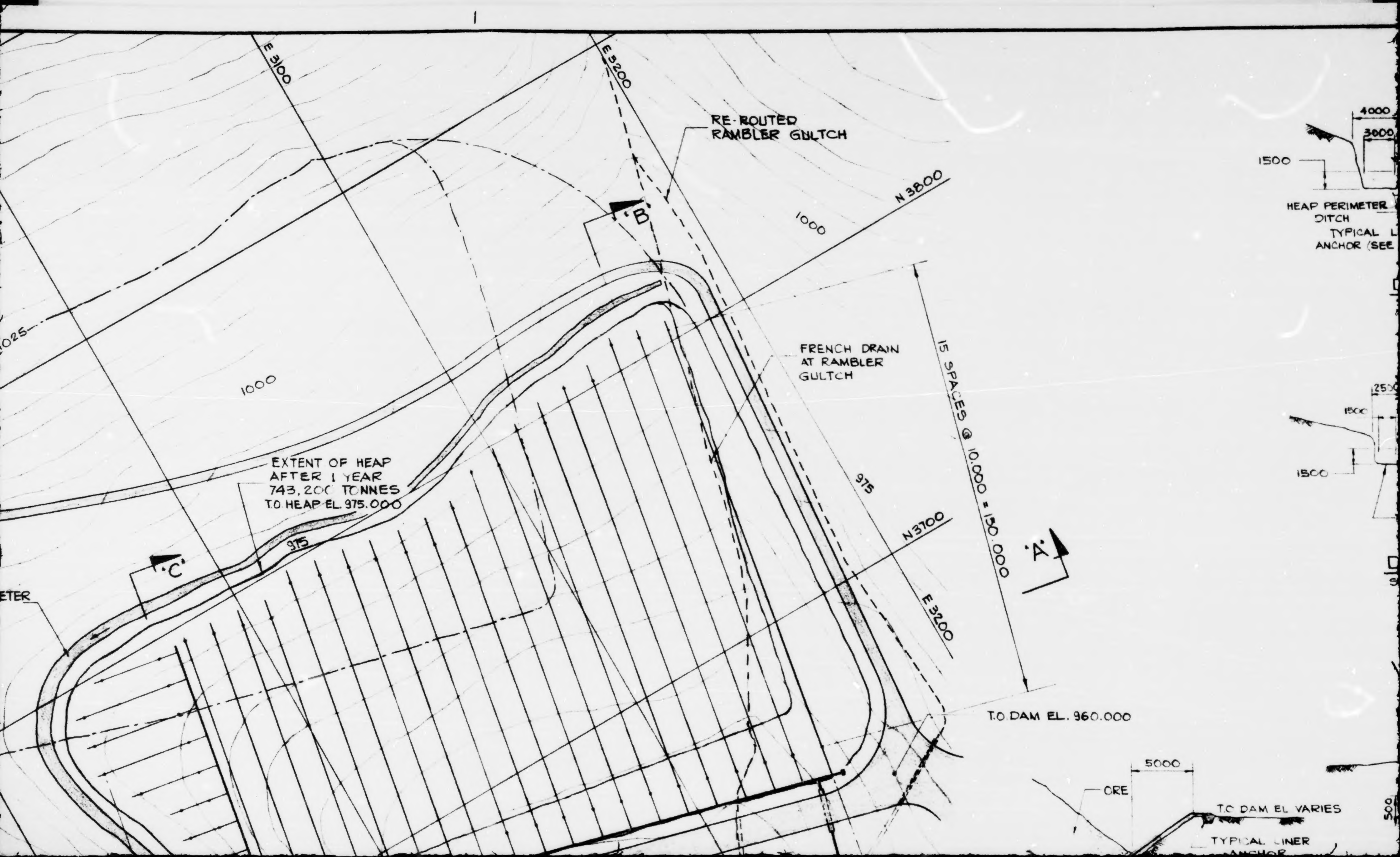
SCALE	1:500 U/N	
DRAWN	E.NATIVIDAD	DEC.19,1986
CHECKED		
APPROVAL		

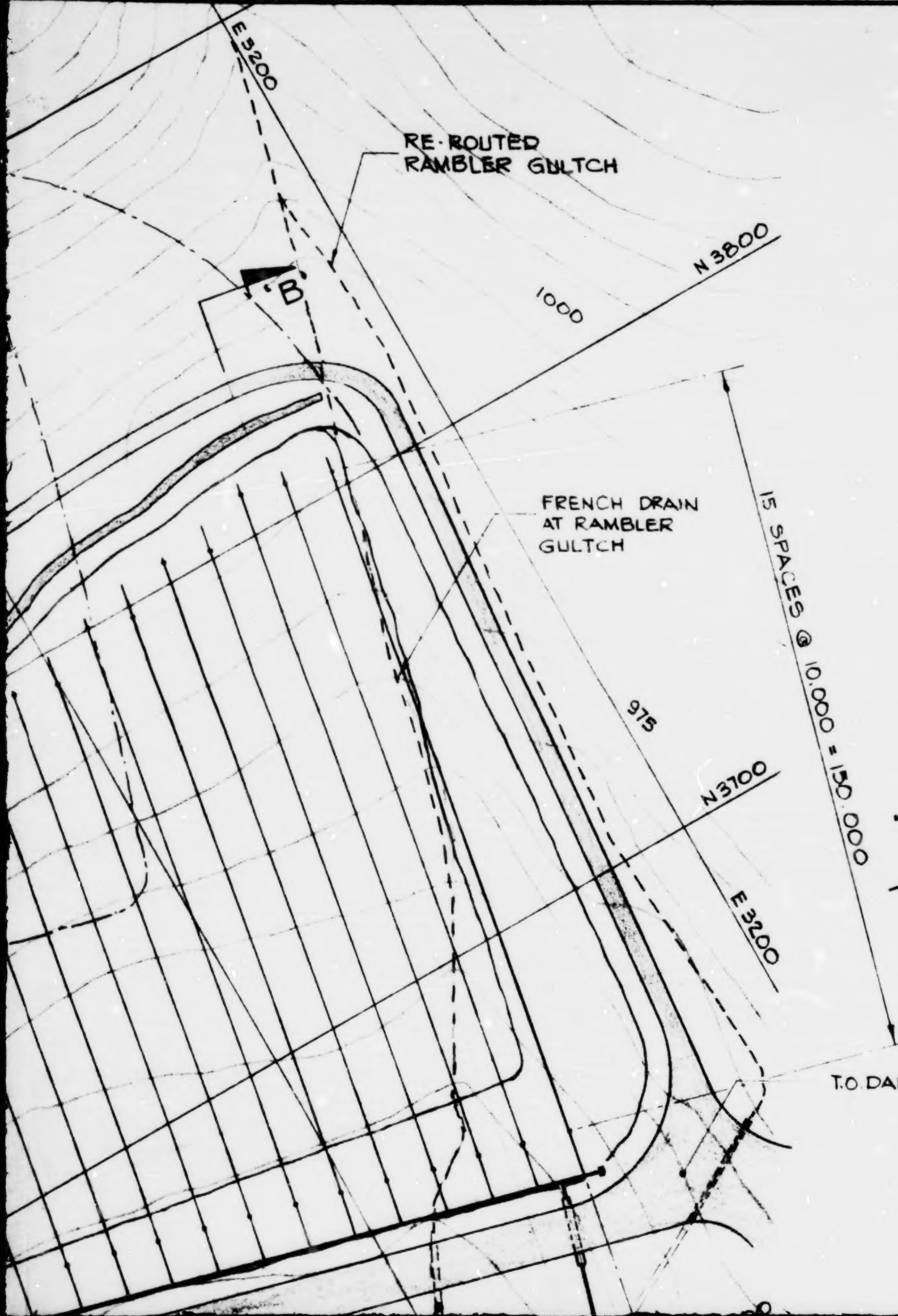
WITTECK DEVELOPMENT INC.

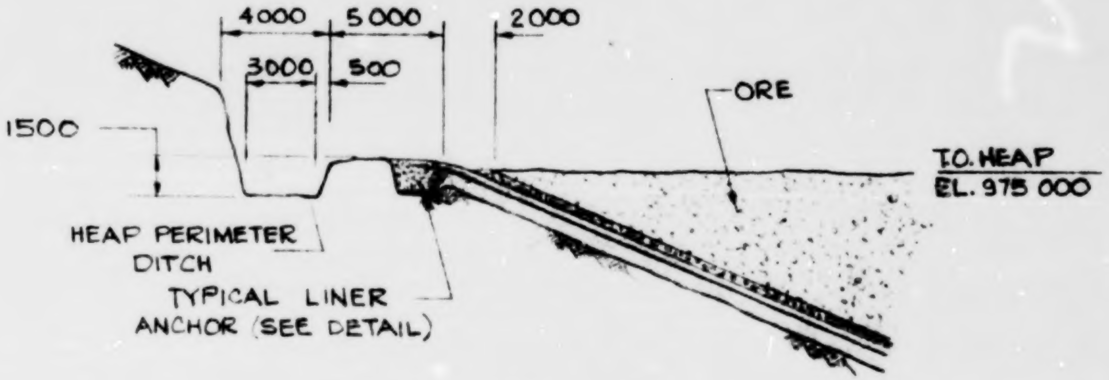
DSS - YUKON HEAP LEACH PROJECT
LEACHING (AREA 30)
BARREN SOLUTION POND GENERAL LAYOUT
PLAN AND SECTIONS

DATE	PROJECT NO.	DWG. No.	REV.
	5209	30 - 130 - 002	P2


1
E 2800
E 2900
E 3000
E 3100
N 4100
N 4000
N 3900
N
1075
1050
1025
1000
975
EXTENT OF HEAP
AFTER 1 YEAR
743,200 TONNES
T.O HEAP EL. 975.000
'C'
HEAP PERIMETER
DITCH
EXTENT OF HEAP
AFTER 5 YEARS
T.O. HEAP EL. 1025.000
NEW ROAD
CRUSHING AREA
GRADE EL. 1020.000

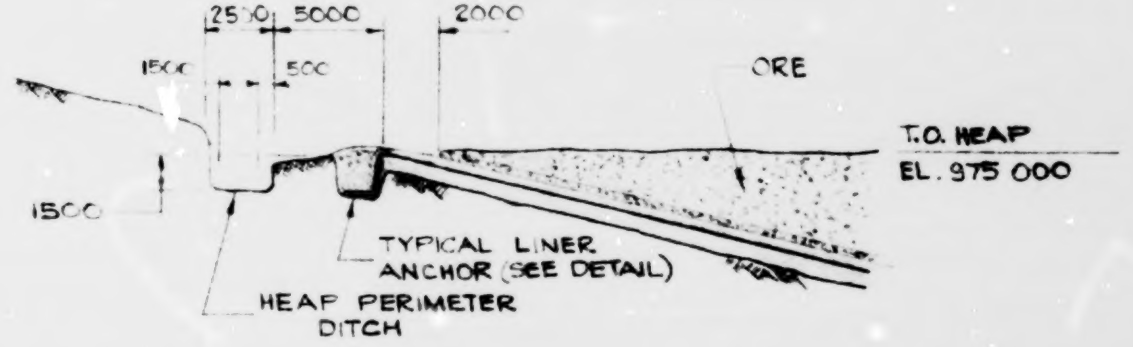

1
E 3100
E 3200
RE-ROUTED
RAMBLER GULTCH
N 3800
1000
'B'
1025
1000
FRENCH DRAIN
AT RAMBLER
GULTCH
15 SPACES @ 10.000 = 150.000
EXTENT OF HEAP
AFTER 1 YEAR
743,200 TONNES
T.O HEAP EL. 975.000
975
975
N 3700
'A'
'C'
ETER
E 3200
T.O DAM EL. 960.000
4000
3000
1500
HEAP PERIMETER
DITCH
TYPICAL L
ANCHOR (SEE
2500
1500
1500
5000
ORE
T.C DAM EL VARIES
TYPICAL LINER
500


E 3200
RE-ROUTED RAMBLER GULTCH
N 3800
1000
'B'
FRENCH DRAIN AT RAMBLER GULTCH
15 SPACES @ 10.000 = 150.000
975
N 3700
'A'
E 3200
T.O DAM EL. 960.000


4000
5000
2000
3000
500
1500
ORE
T.O. HEAP
EL. 975 000
HEAP PERIMETER DITCH
TYPICAL LINER ANCHOR (SEE DETAIL)

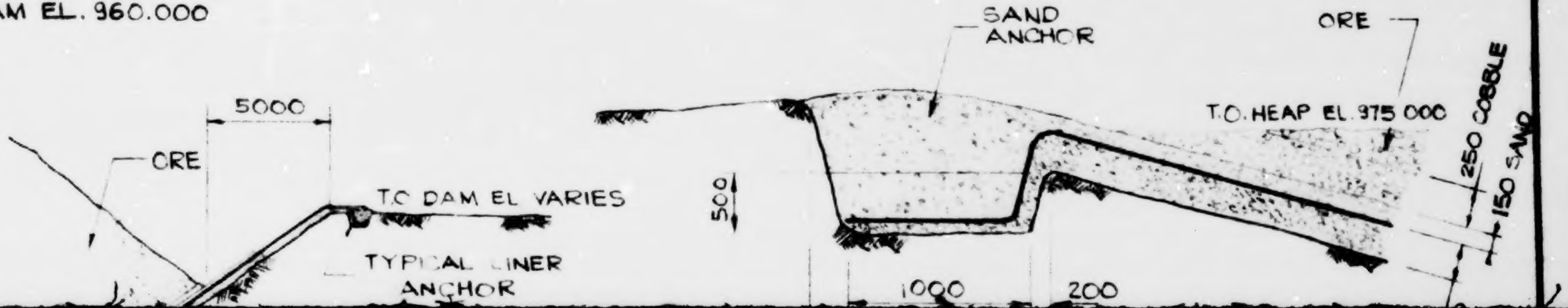
DETAIL 'D'

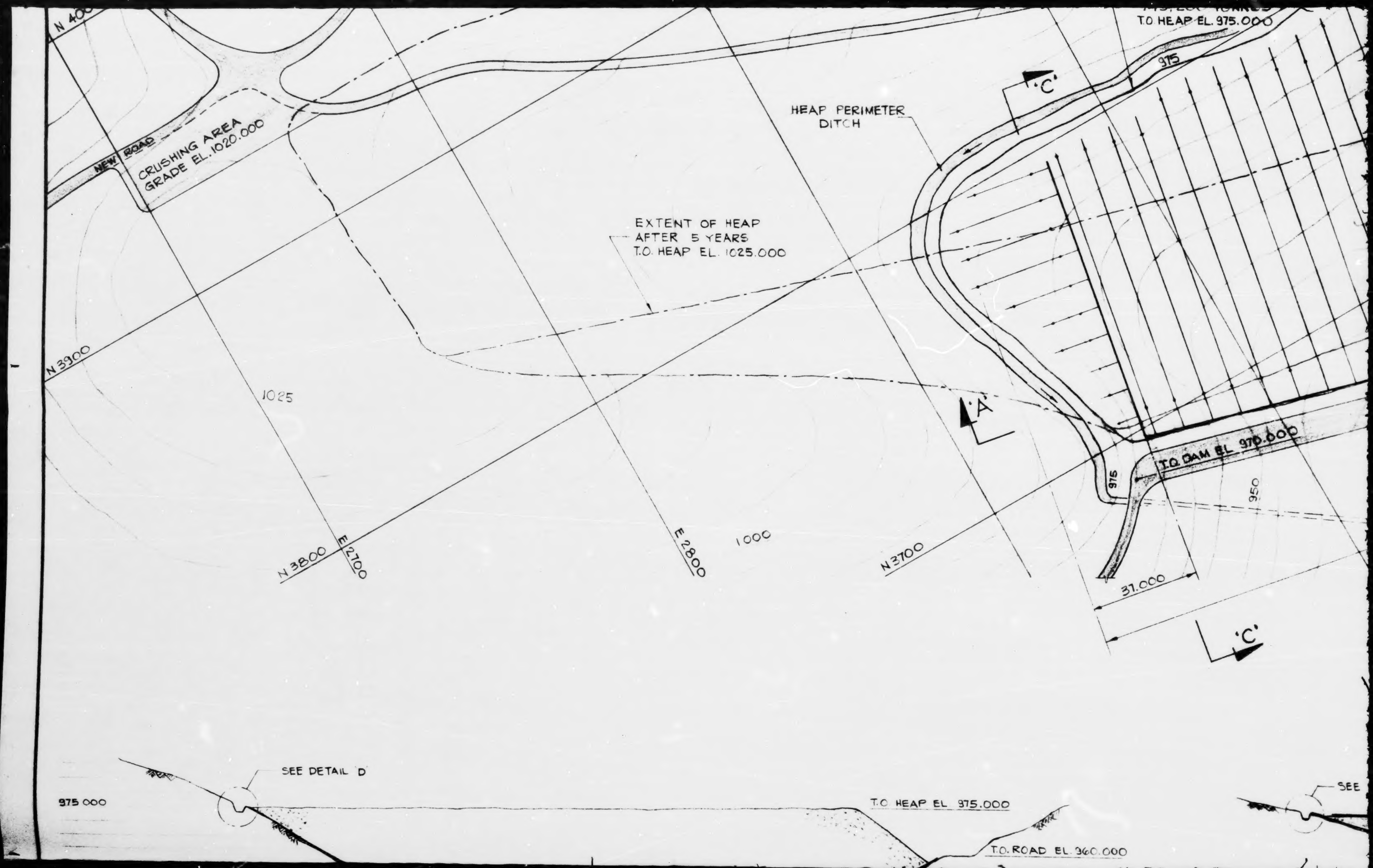
SCALE = 1:250

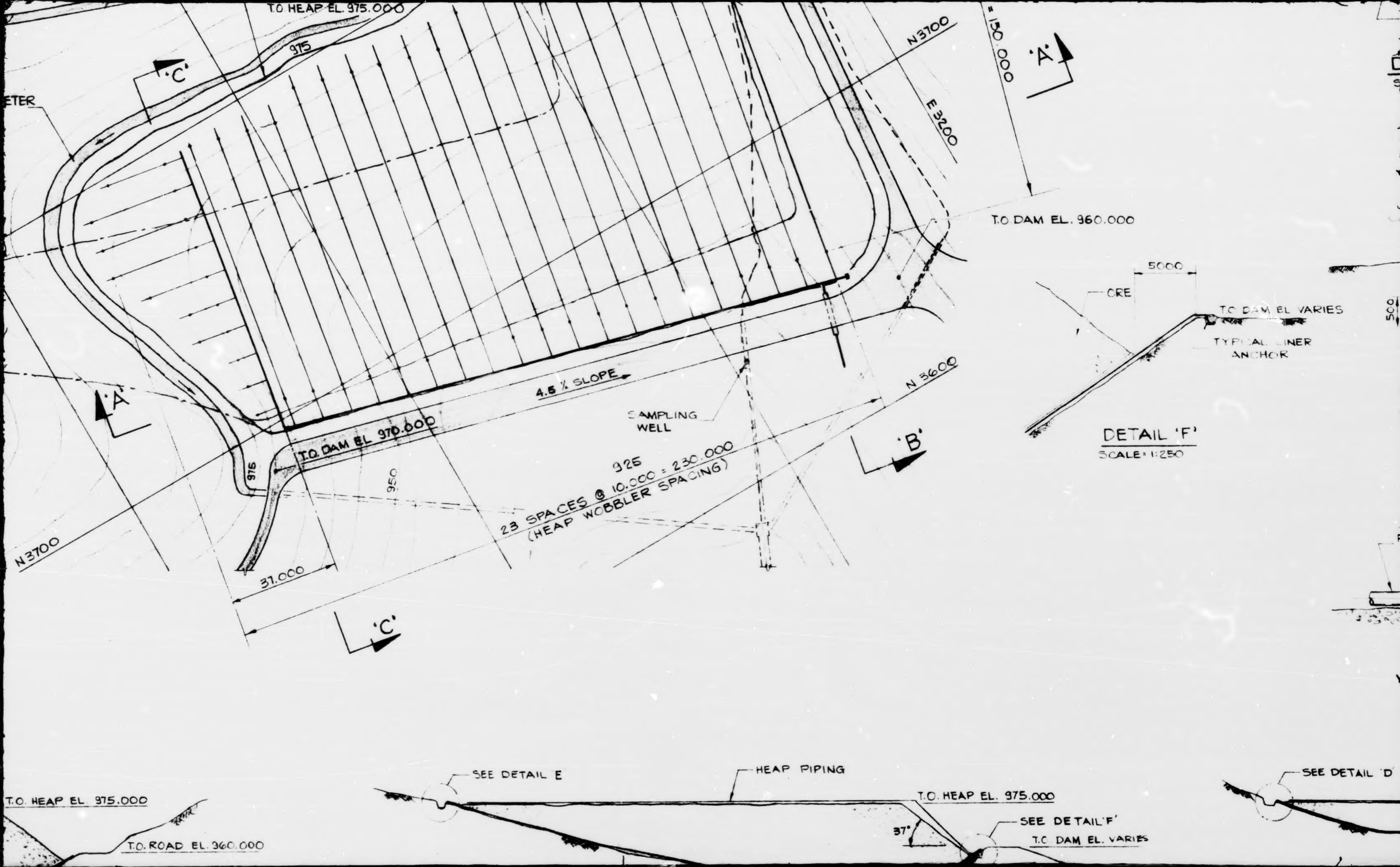

2500
5000
2000
1500
500
ORE
T.O. HEAP
EL. 975 000
1500
TYPICAL LINER ANCHOR (SEE DETAIL)
HEAP PERIMETER DITCH

DETAIL 'E'

SCALE = 1:250


5000
ORE
T.O DAM EL VARIES
TYPICAL LINER ANCHOR
SAND ANCHOR
ORE
T.O. HEAP EL. 975 000
250 COBBLE
150 SAND
500
1000
200


N 400
T.O HEAP EL. 975.000
975
'C'
HEAP PERIMETER
DITCH
NEW ROAD
CRUSHING AREA
GRADE EL. 1020.000
EXTENT OF HEAP
AFTER 5 YEARS
T.O. HEAP EL. 1025.000
N 3900
1025
'A'
T.O. DAM EL 970.000
975
950
N 3800
E 2700
E 2800
1000
N 3700
37.000
'C'
SEE DETAIL 'D'
975.000
T.O HEAP EL 975.000
T.O. ROAD EL. 960.000
SEE


T.O. HEAP EL. 975.000
975
'C'
ETER
N3700
E3200
'A'
=150.000
T.O DAM EL. 960.000
4.5 % SLOPE
N 3600
SAMPLING WELL
T.O. DAM EL 970.000
'B'
925
950
975
23 SPACES @ 10.000 = 230.000
(HEAP WOBBLER SPACING)
N3700
31.000
'C'
'A'
5000
ORE
T.O DAM EL VARIES
TYPICAL LINER ANCHOR
500
SEE DETAIL E
HEAP PIPING
T.O. HEAP EL. 975.000
SEE DETAIL 'D'
SEE DETAIL 'F'
T.C DAM EL. VARIES
37°
T.O. HEAP EL. 975.000
T.O. ROAD EL. 960.000

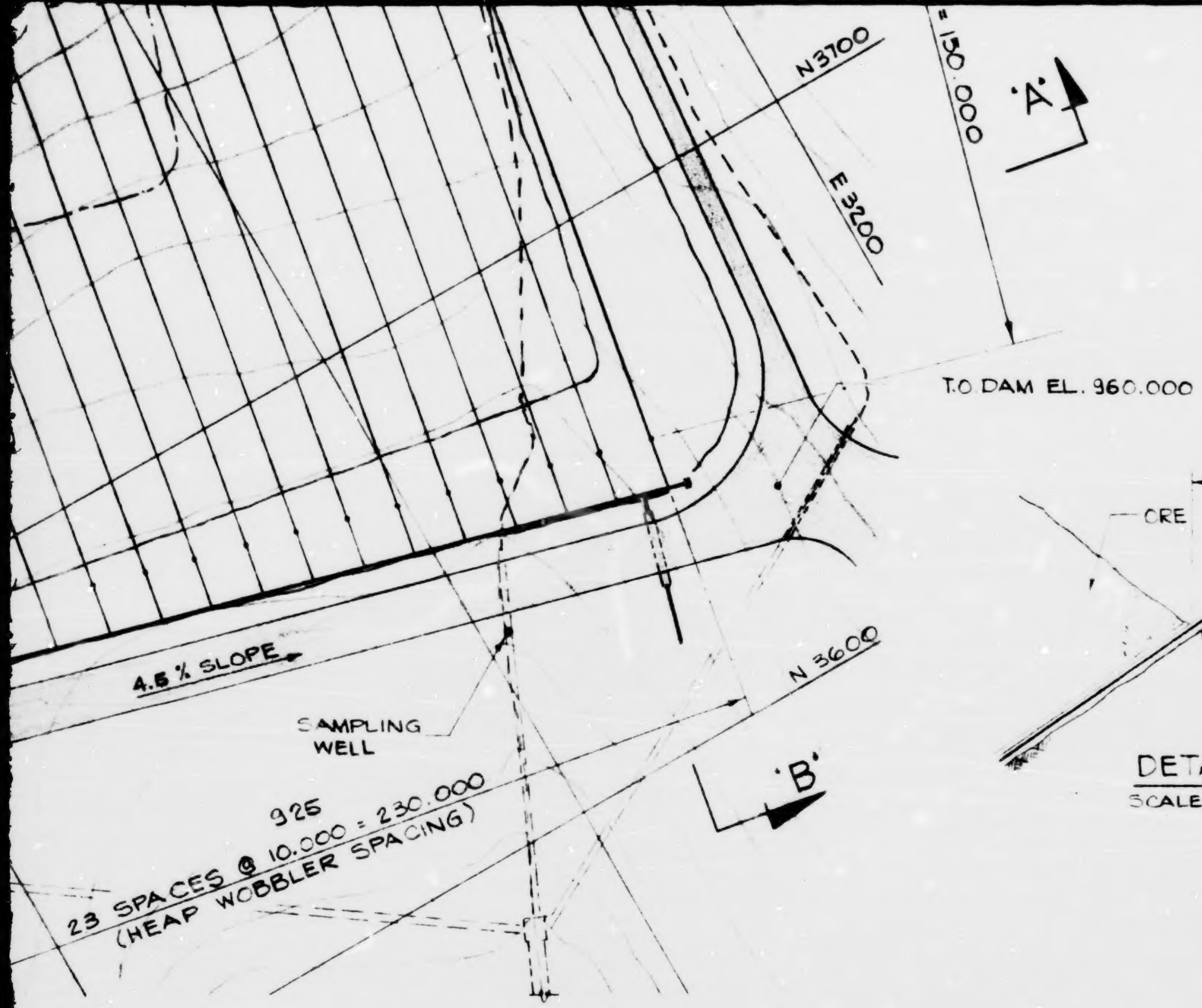
DETAIL 'F'

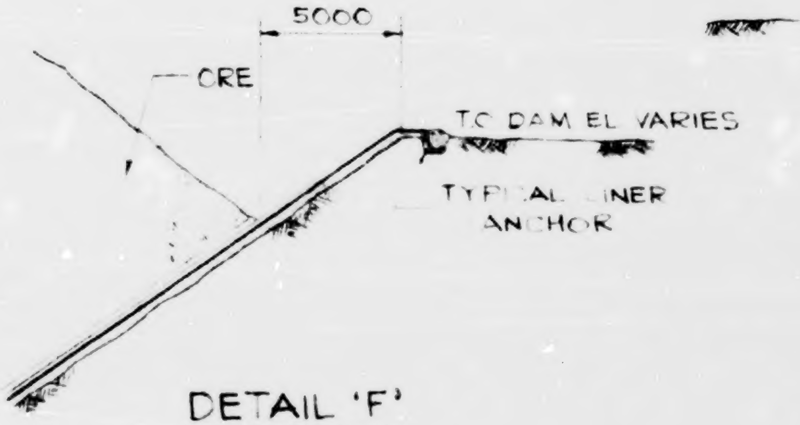
SCALE = 1:250

HEAP PERIMETER DITCH

DETAIL 'E'

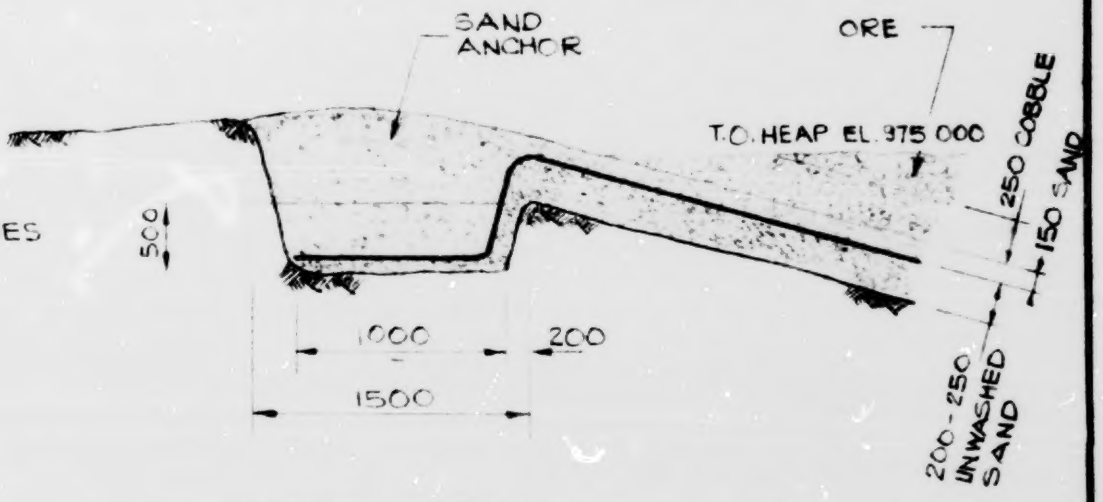
SCALE = 1:250


N3700
150.000
'A'
E3200
T.O DAM EL. 960.000
4.5 % SLOPE
N 3600
SAMPLING WELL
'B'
925
23 SPACES @ 10.000 = 230.000
(HEAP WOBBLER SPACING)


5000
ORE
T.O DAM EL VARIES
TYPICAL LINER ANCHOR

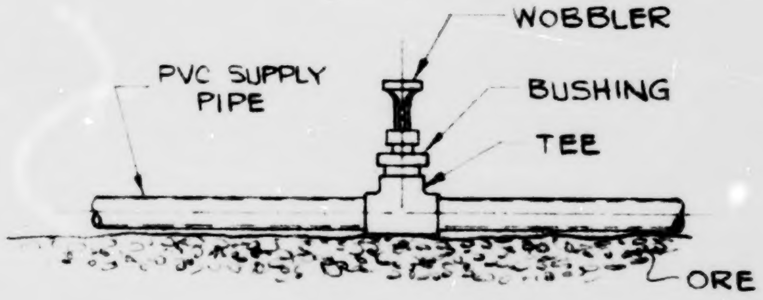
DETAIL 'F'

SCALE = 1:250


SAND ANCHOR
ORE
T.O. HEAP EL. 975.000
250 COBBLE
150 SAND
500
1000
200
1500
200-250 UNWASHED SAND

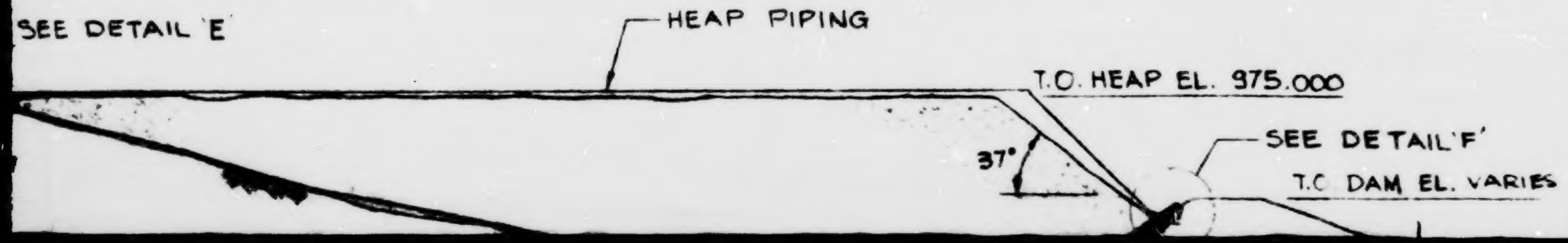
ANCHOR DETAIL

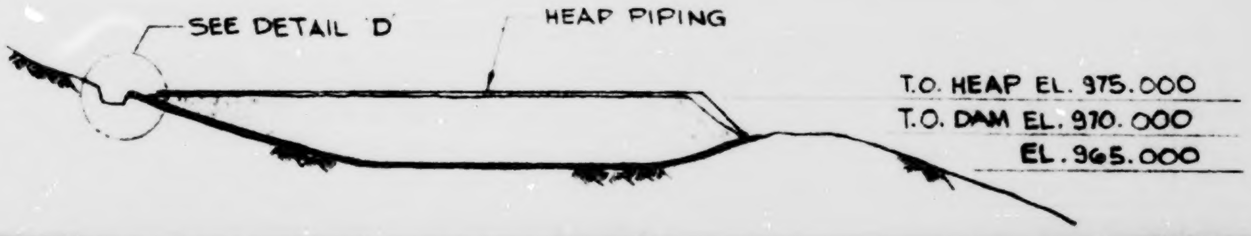
SCALE = 1:50

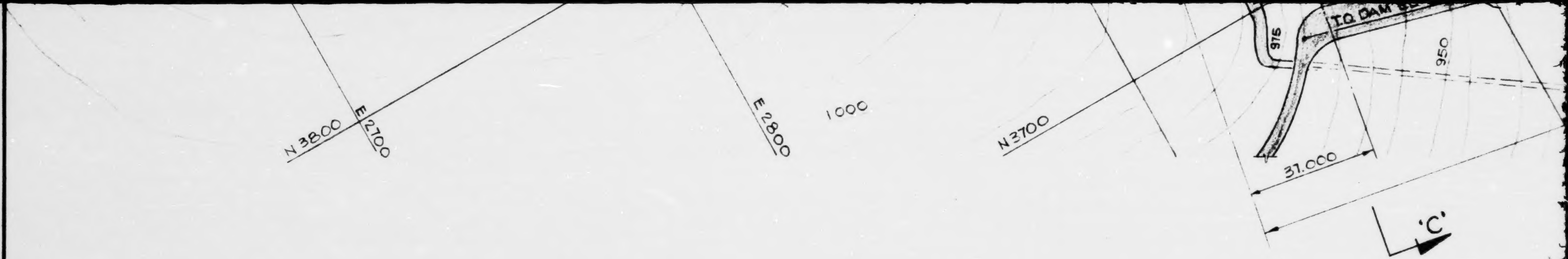

WOBBLER
PVC SUPPLY PIPE
BUSHING
TEE
ORE

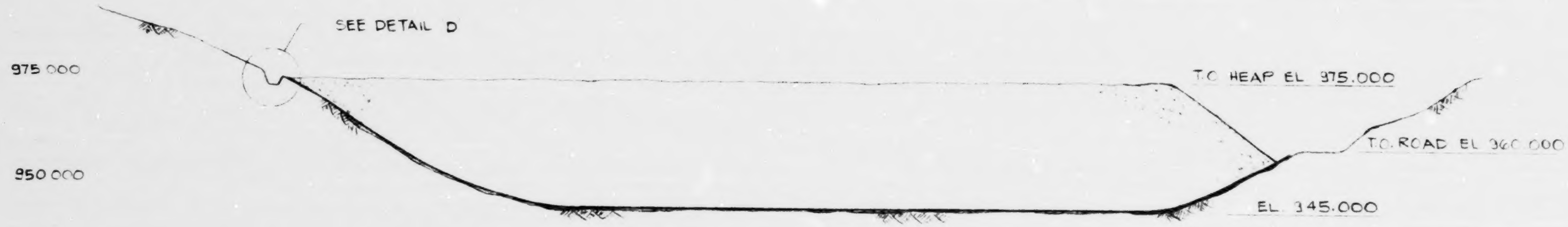
TYPICAL WOBBLER INSTALLATION

SCALE = 1:10


SEE DETAIL 'E'
HEAP PIPING
T.O. HEAP EL. 975.000
37°
SEE DETAIL 'F'
T.O DAM EL. VARIES


SEE DETAIL 'D'
HEAP PIPING
T.O. HEAP EL. 975.000
T.O. DAM EL. 970.000
EL. 965.000


N3800
E2700
E2800
1000
N3700
975
T.O. DAM EL
950
37.000
'C'

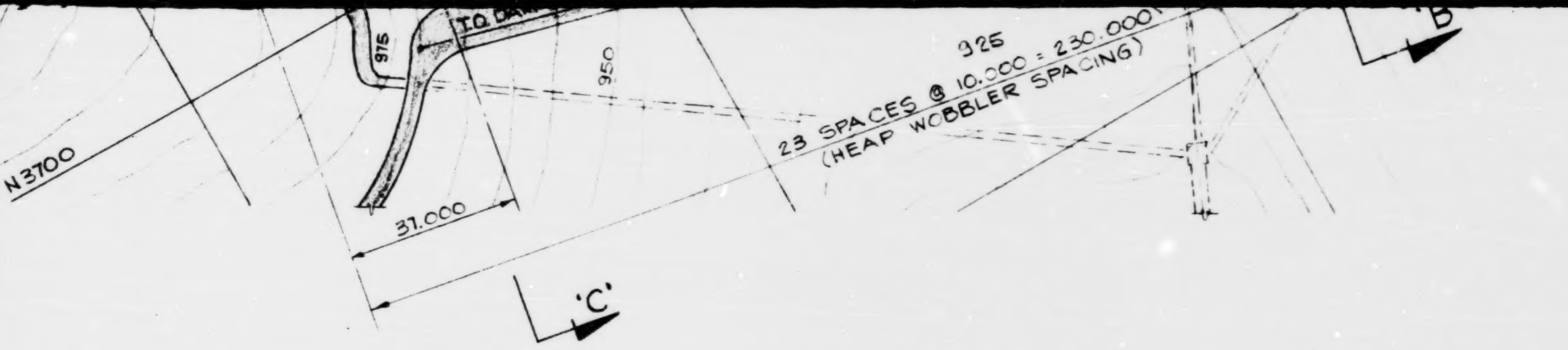

SEE DETAIL D
975.000
950.000
T.O. HEAP EL 975.000
T.O. ROAD EL 960.000
EL 945.000

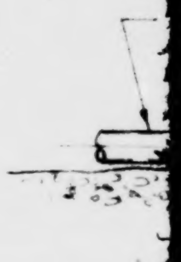
SECTION A-A

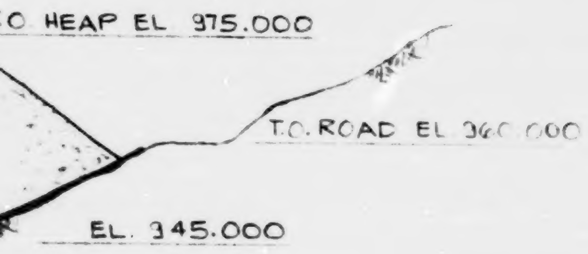
DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHE
40-130-002	PROCESS PLANT LAYOUT - SECTION							
40-130-001	PROCESS PLANT LAYOUT - PLAN							
10-120-002	LOCAL SITE LAYOUT							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND							
10-120-001	GENERAL SITE PLAN							
10-110-002	FLOW DIAGRAM (SHEET 2 OF 2)							
10-110-001	FLOW DIAGRAM (SHEET 1 OF 2)							

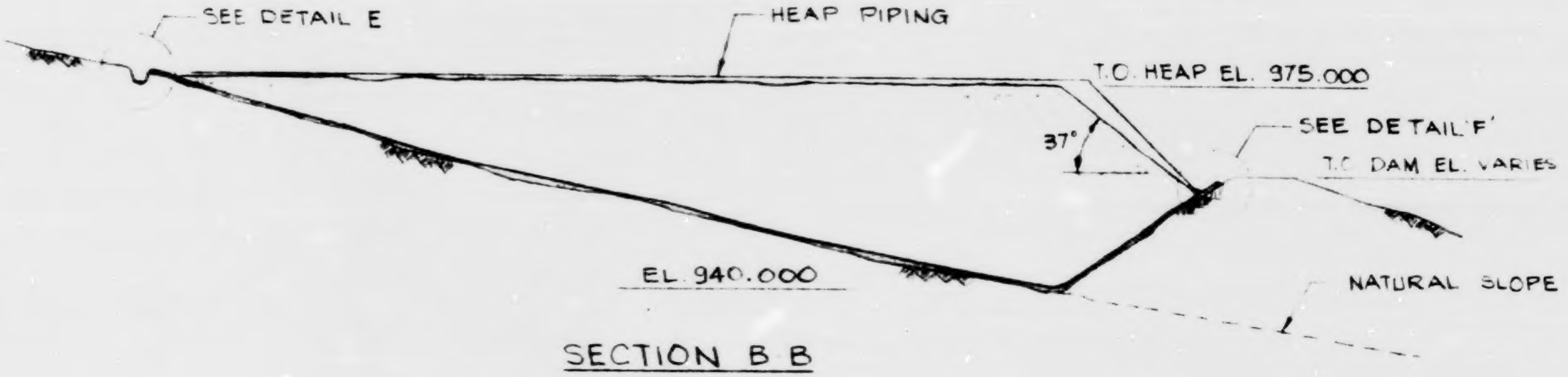
DETAIL

SCALE: 1:250


T.O. DAM
975
950
925
23 SPACES @ 10.000 = 230.000
(HEAP WOBBLER SPACING)
N3700
37.000
'C'
B




O HEAP EL. 975.000
T.O. ROAD EL. 360.000
EL. 345.000


SEE DETAIL E
HEAP PIPING
T.O. HEAP EL. 975.000
SEE DETAIL 'F'
T.O. DAM EL. VARIES
37°
EL. 940.000
NATURAL SLOPE

SECTION B B

SEE DETAIL 'D'

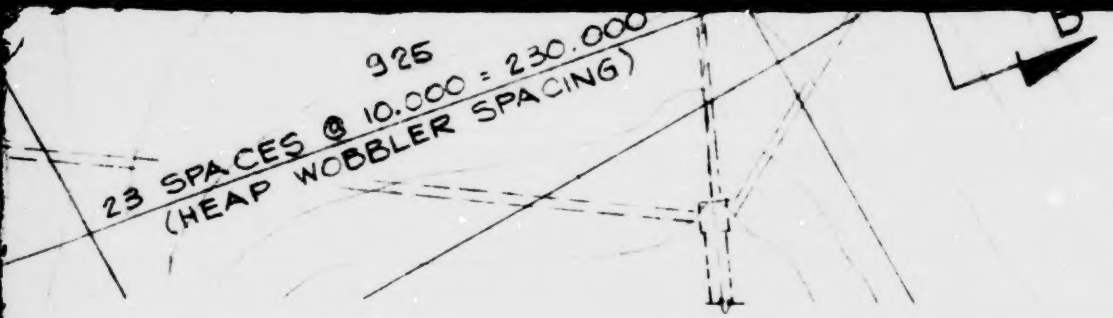
SECTIO

No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
					P2	MAY 20,87	RELEAED FOR REPORT	J. LI	
					PI		RELEASED FOR INFORMATION	EN	

SCALE	1:1000 4/N	
DRAWN	E.NATIVIDAD	17 DEC '86
CHECKED		
APPROVAL		

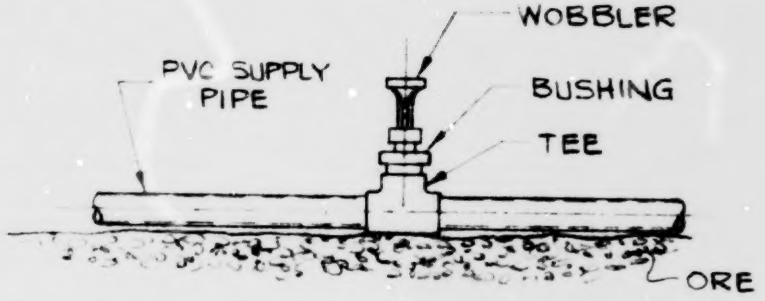
DSS

DATE

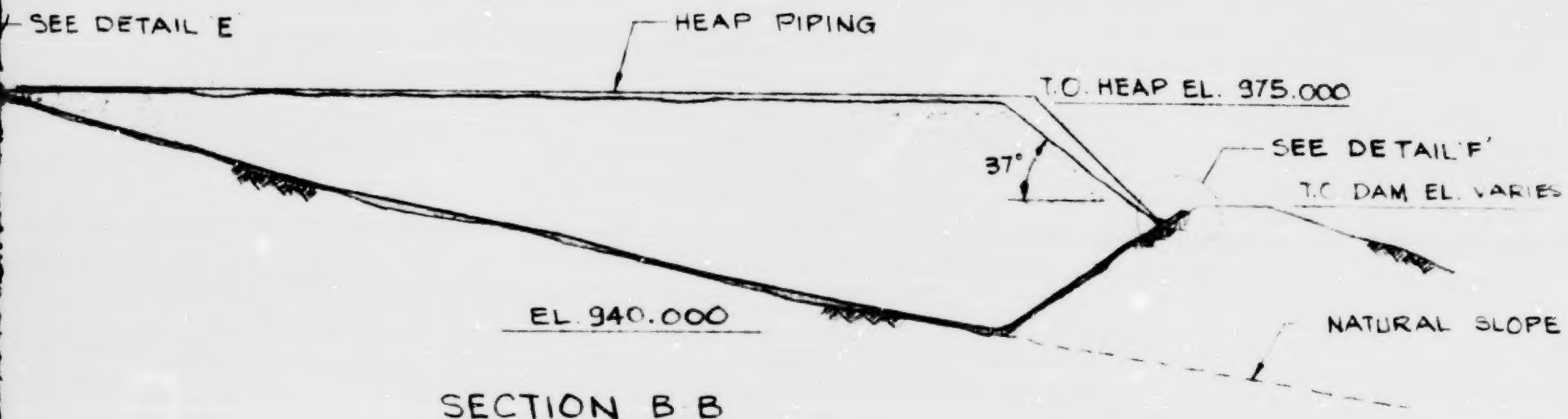

925
23 SPACES @ 10.000 = 230.000
(HEAP WOBBLER SPACING)
B

SCALE=1:250

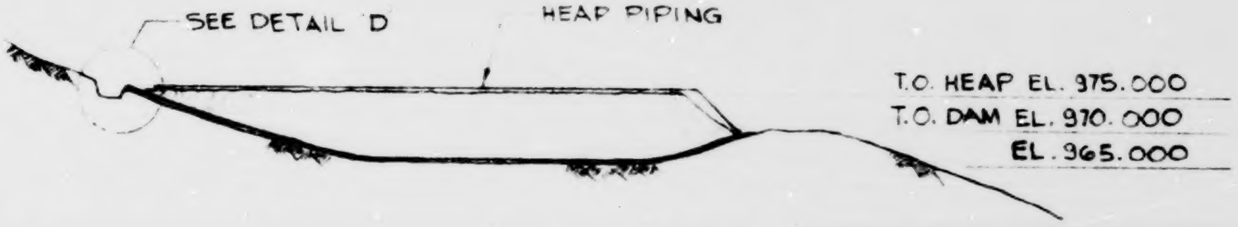
SCALE = 1:50


WOBBLER
PVC SUPPLY PIPE
BUSHING
TEE
ORE

TYPICAL WOBBLER INSTALLATION
SCALE = 1:10


SEE DETAIL E
HEAP PIPING
T.O. HEAP EL. 975.000
37°
SEE DETAIL 'F'
T.O. DAM EL. VARIES
EL. 940.000
NATURAL SLOPE

SECTION B-B


SEE DETAIL D
HEAP PIPING
T.O. HEAP EL. 975.000
T.O. DAM EL. 970.000
EL. 965.000

SECTION C-C

CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEAED FOR REPORT	J. LI	
	P1		RELEASED FOR INFORMATION	EN	

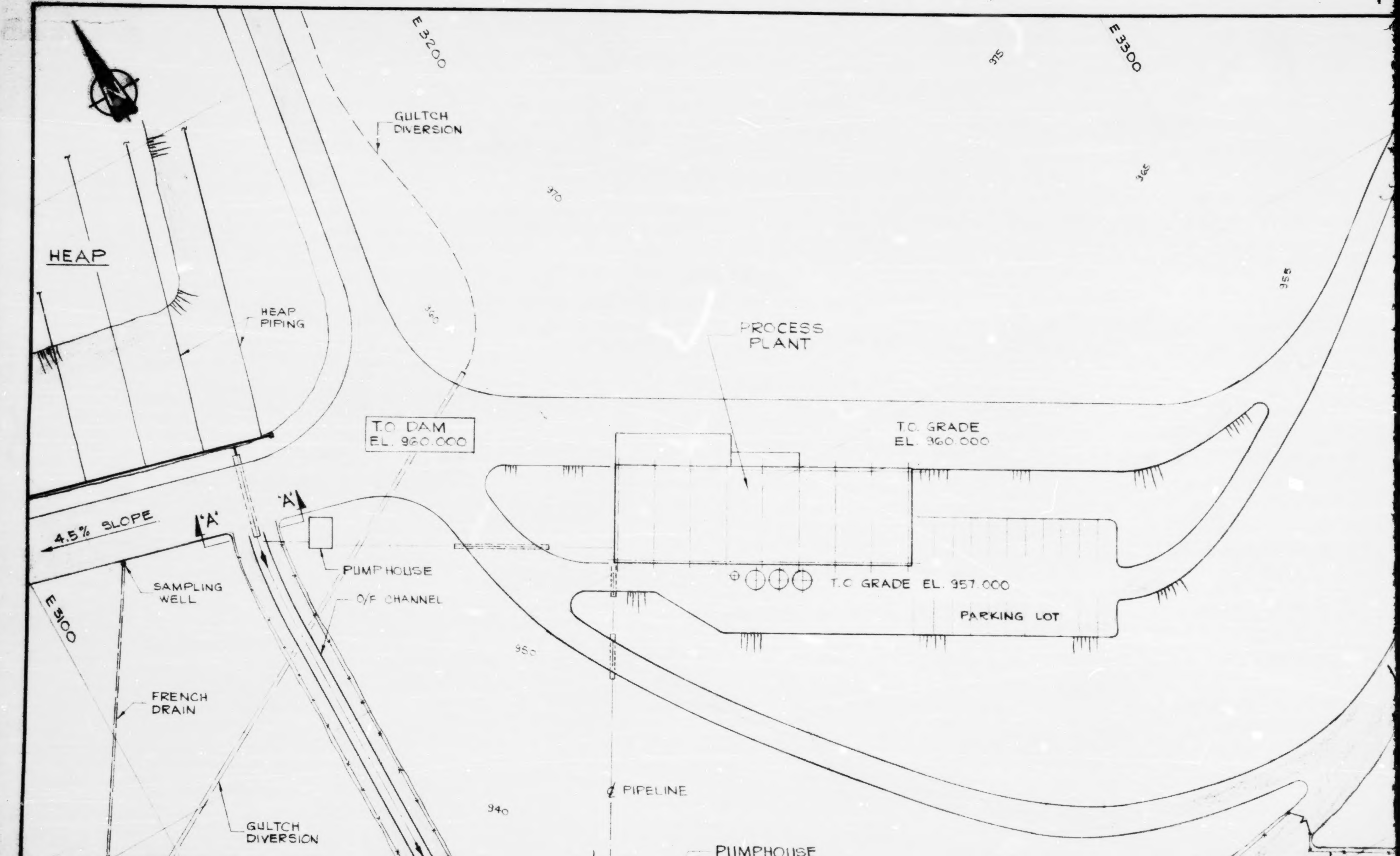
SCALE	1:1000 ⁴⁄ₙ	
DRAWN	E. NATIVIDAD	17 DEC '86
CHECKED		
APPROVAL		

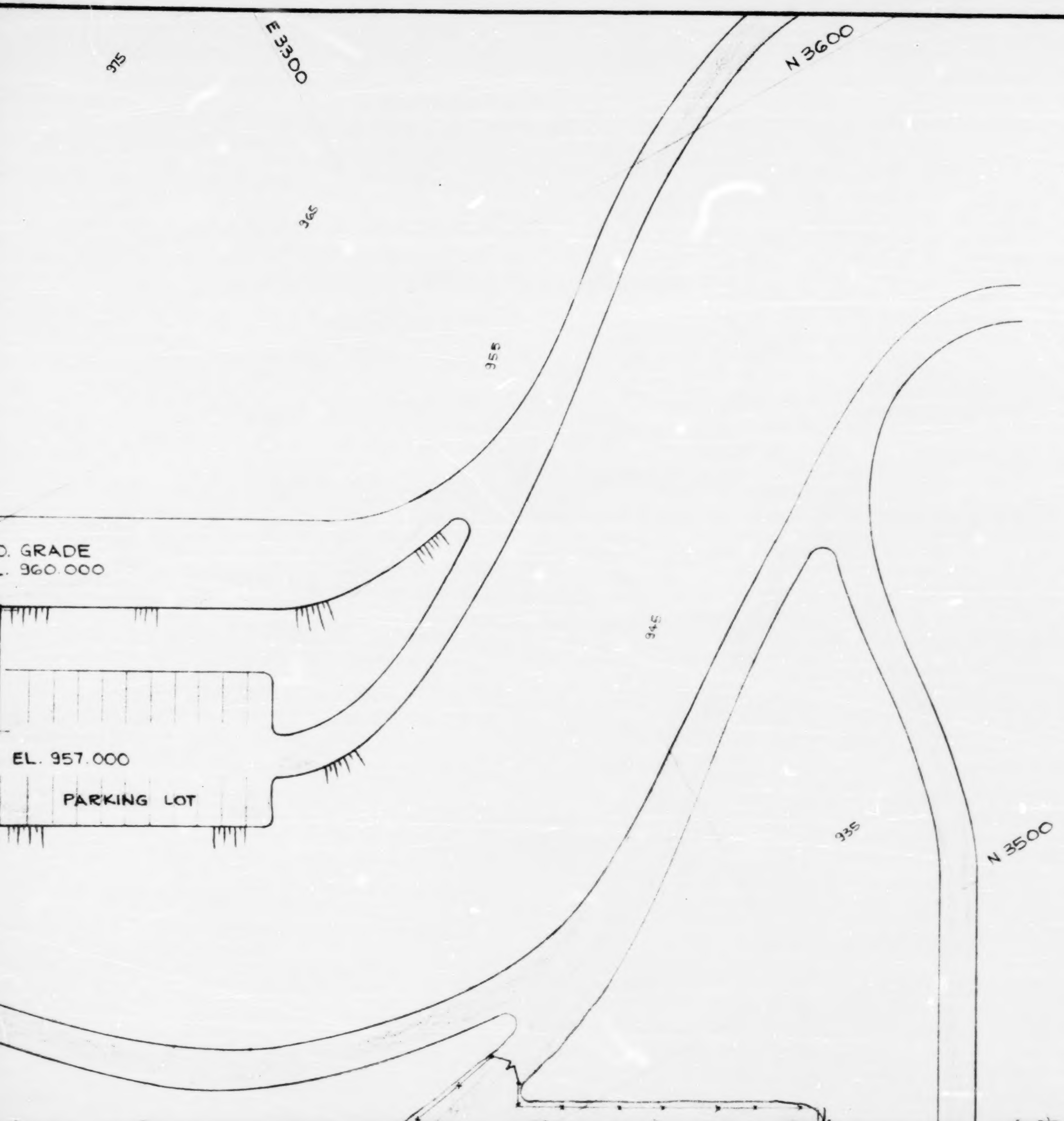
WDI

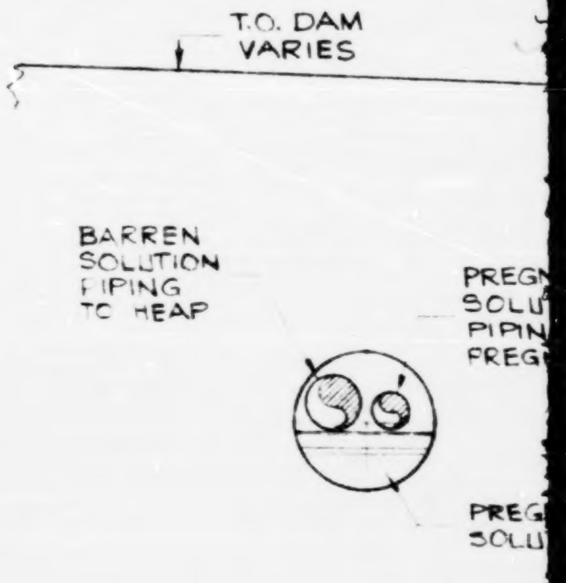
WITTECK DEVELOPMENT INC.

DSS - YUKON HEAP LEACH PROJECT
LEACHING (AREA 30)
HEAP GENERAL LAYOUT
PLAN AND SECTIONS

DATE	PROJECT NO.	DWG. No.	REV.
	5209	30 - 130 - 001	P2

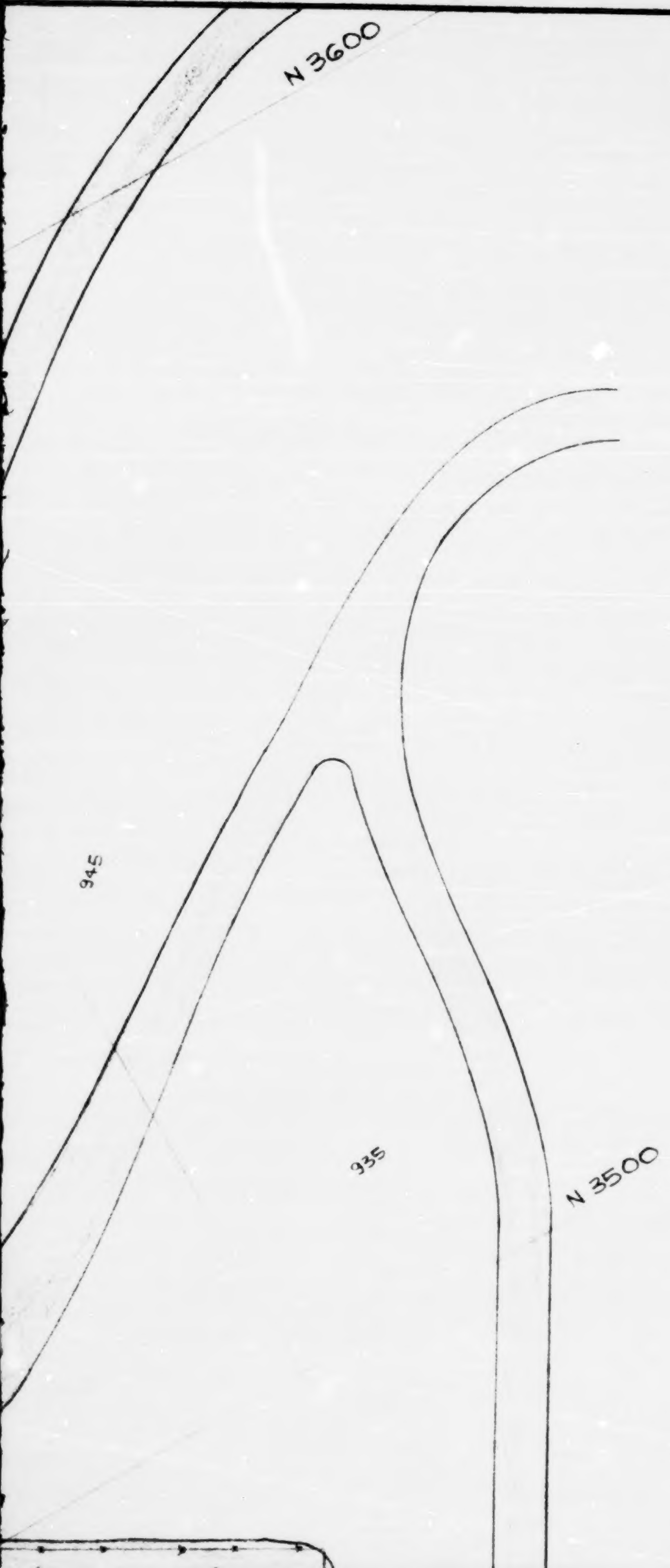

E 3200
E 3300
375
GULTCH
DIVERSION
970
365
HEAP
HEAP
PIPING
955
960
PROCESS
PLANT
T.O. DAM
EL. 960.000
T.O. GRADE
EL. 960.000
'A'
4.5% SLOPE
'A'
PUMP HOUSE
T.O. GRADE EL. 957.000
SAMPLING
WELL
O/F CHANNEL
PARKING LOT
E 3100
950
FRENCH
DRAIN
℄ PIPELINE
940
GULTCH
DIVERSION
PUMPHOUSE

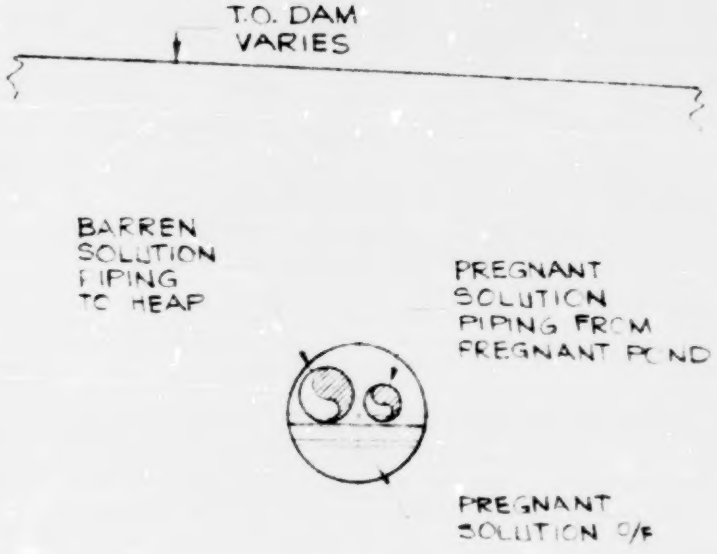

375
E 3300
N 3600
365
355
945
935
N 3500
D. GRADE
. 960.000
EL. 957.000
PARKING LOT


T.O. DAM
VARIES
BARREN
SOLUTION
PIPING
TO HEAP
PREG
SOLU
PIPIN
PREG
PREG
SOLU

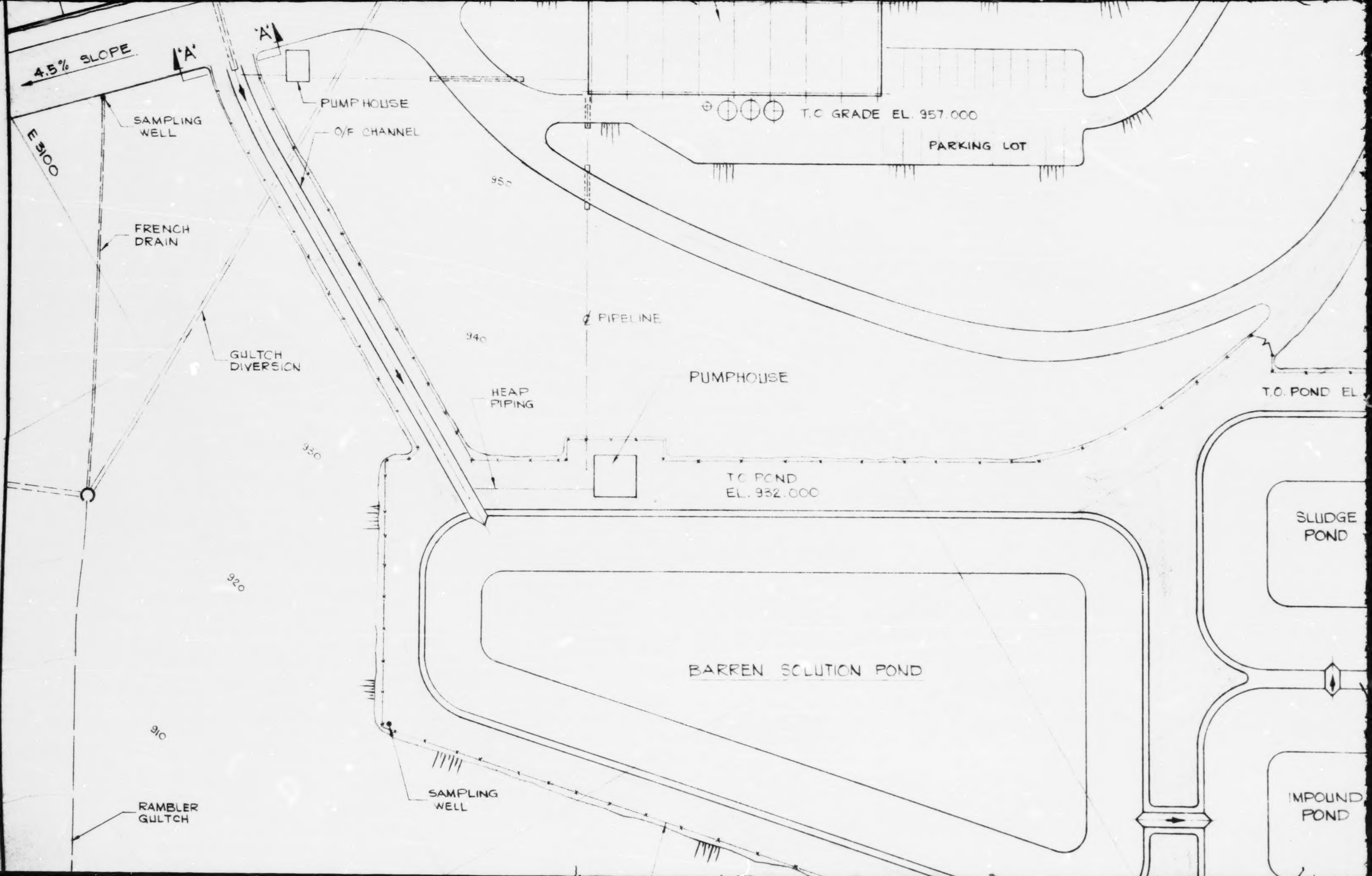
SECTION 'A-

SCALE 1:50


N 3600
945
935
N 3500

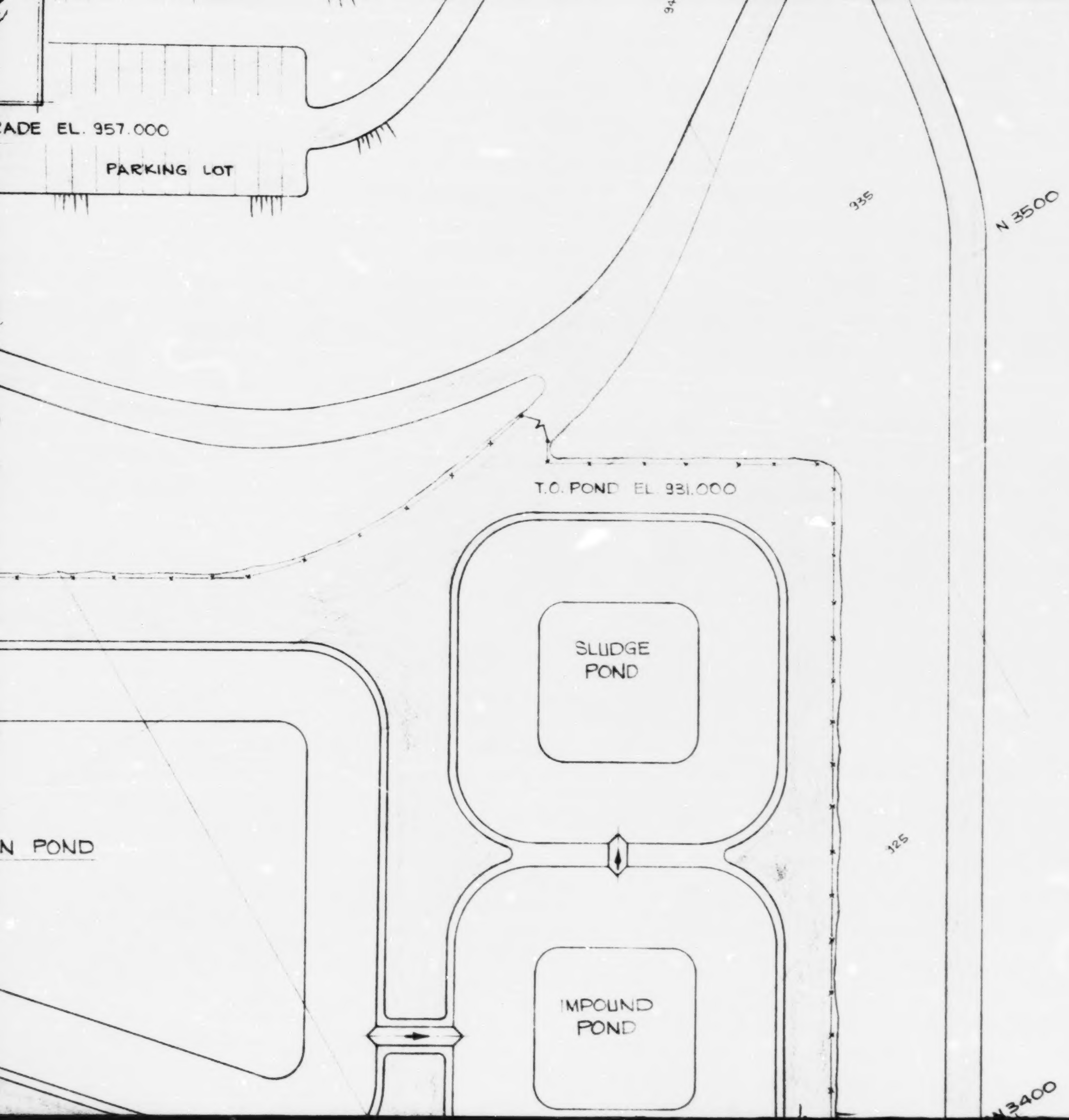

T.O. DAM
VARIES
BARREN
SOLUTION
PIPING
TO HEAP
PREGNANT
SOLUTION
PIPING FROM
PREGNANT POND
PREGNANT
SOLUTION P/F

SECTION 'A-A'
SCALE 1:50

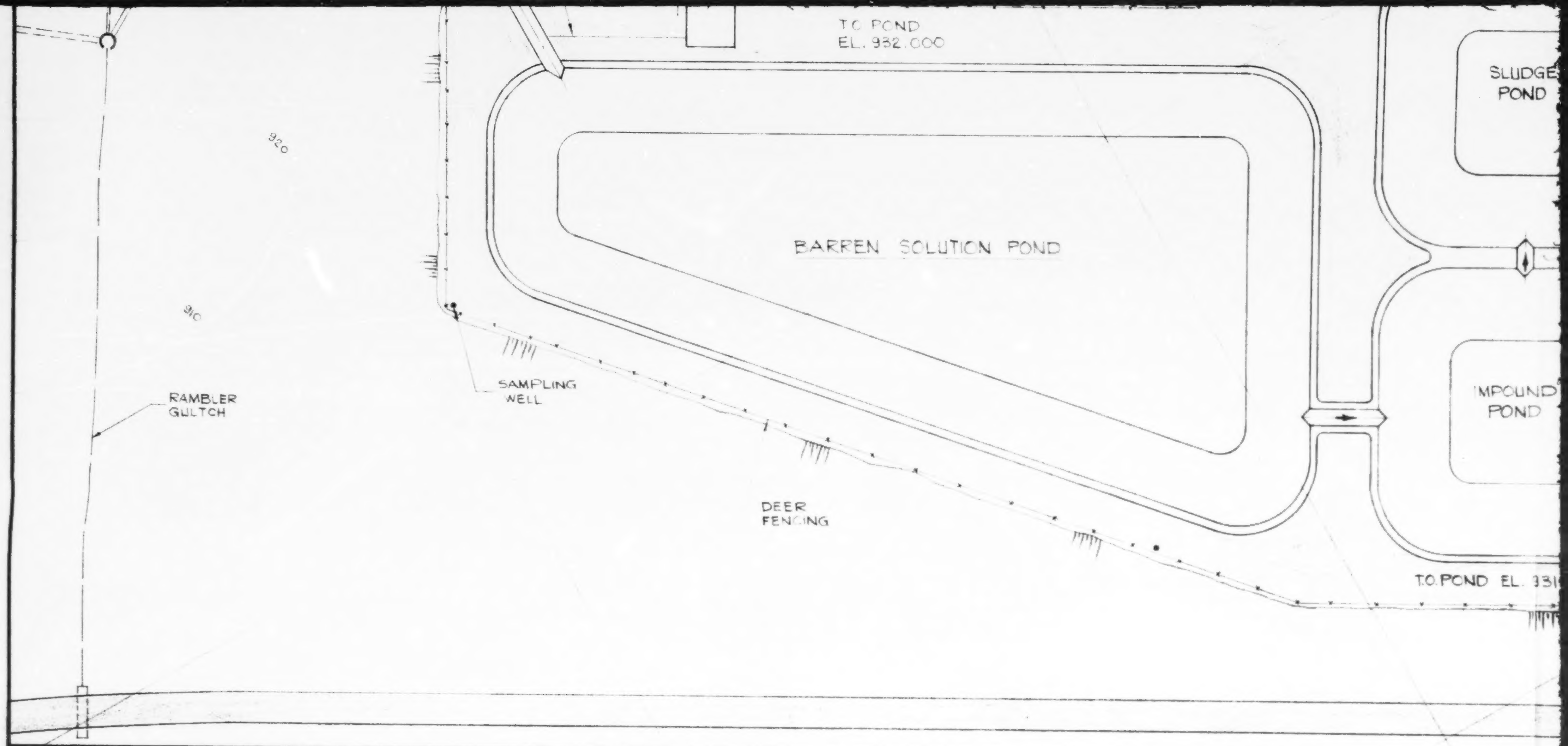

4.5% SLOPE
'A'
'A'
PUMPHOUSE
SAMPLING WELL
O/F CHANNEL
E 3100
T.O GRADE EL. 957.000
PARKING LOT
950
FRENCH DRAIN
℄ PIPELINE
940
GULTCH DIVERSION
PUMPHOUSE
T.O. POND EL
HEAP PIPING
930
T.O POND EL. 932.000
SLUDGE POND
920
BARREN SOLUTION POND
910
SAMPLING WELL
RAMBLER GULTCH
IMPOUND POND

SECTION

SCALE 1 50

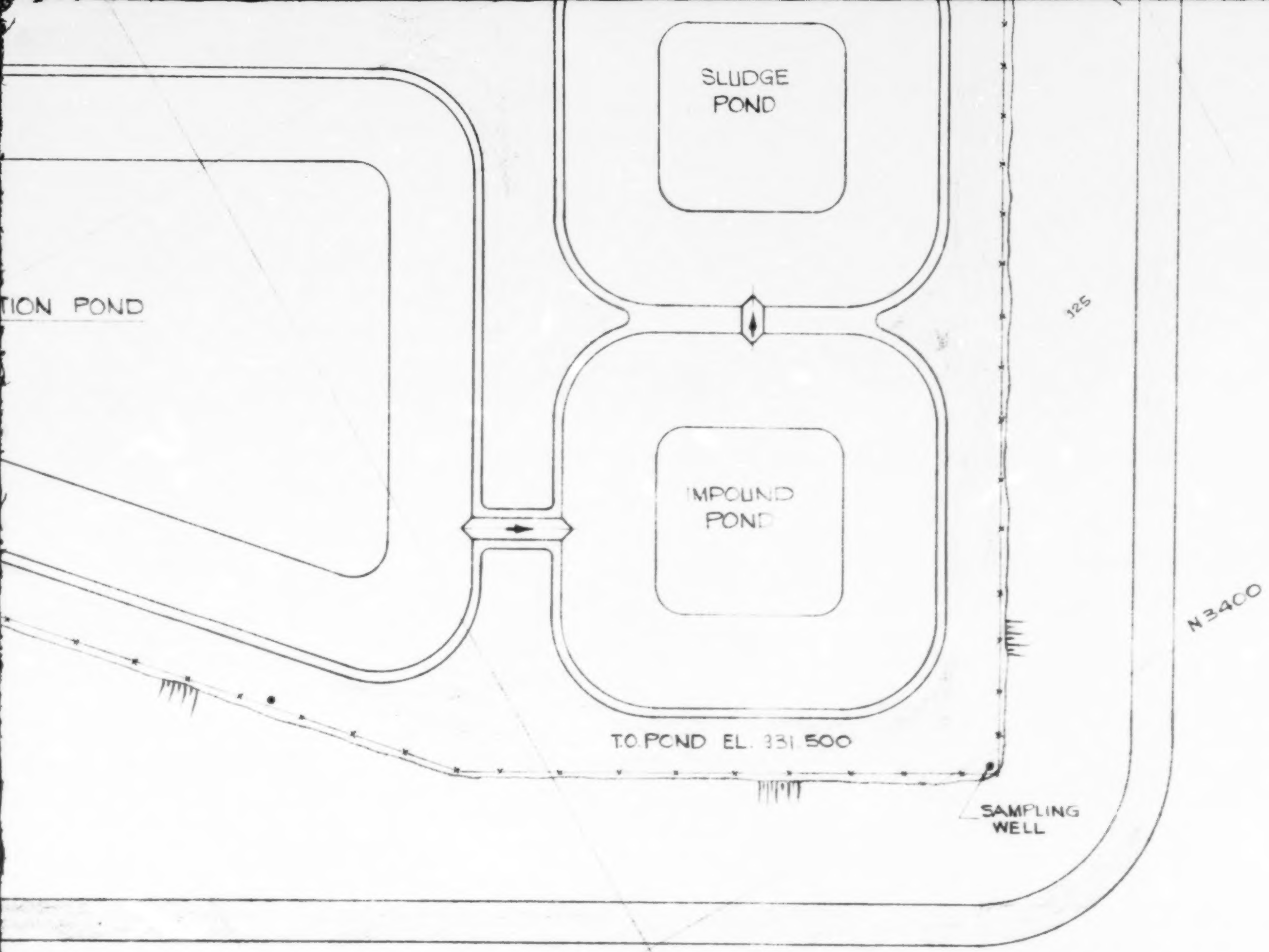

94
ADE EL. 957.000
PARKING LOT
935
N 3500
T.O. POND EL. 931.000
SLUDGE POND
N POND
925
IMPOUND POND
N 3400

SECTION 'A-A'
SCALE 1:50
935
N 3500
ND EL. 931.000
UDGE
OND
925
OUND
OND
N3400


T.O. POND
EL. 932.000
SLUDGE
POND
920
BARREN SOLUTION POND
910
SAMPLING
WELL
RAMBLER
GULTCH
IMPOUND
POND
DEER
FENCING
T.O. POND EL. 931

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHEC
10-130-002	PROCESS PLANT LAYOUT - SECTION							
10-130-001	PROCESS PLANT LAYOUT - PLAN							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND GENERAL LAYOUT							
30-130-001	HEAP GENERAL LAYOUT							
10-120-001	GENERAL SITE PLAN							
10-110-002	FLOW DIAGRAM (SHEET 2 OF 2)							
10-110-001	FLOW DIAGRAM (SHEET 1 OF 2)							

TRANSTEX


SLUDGE POND
TION POND
325
IMPOUND POND
N3400
T.O.POND EL. 331.500
SAMPLING WELL

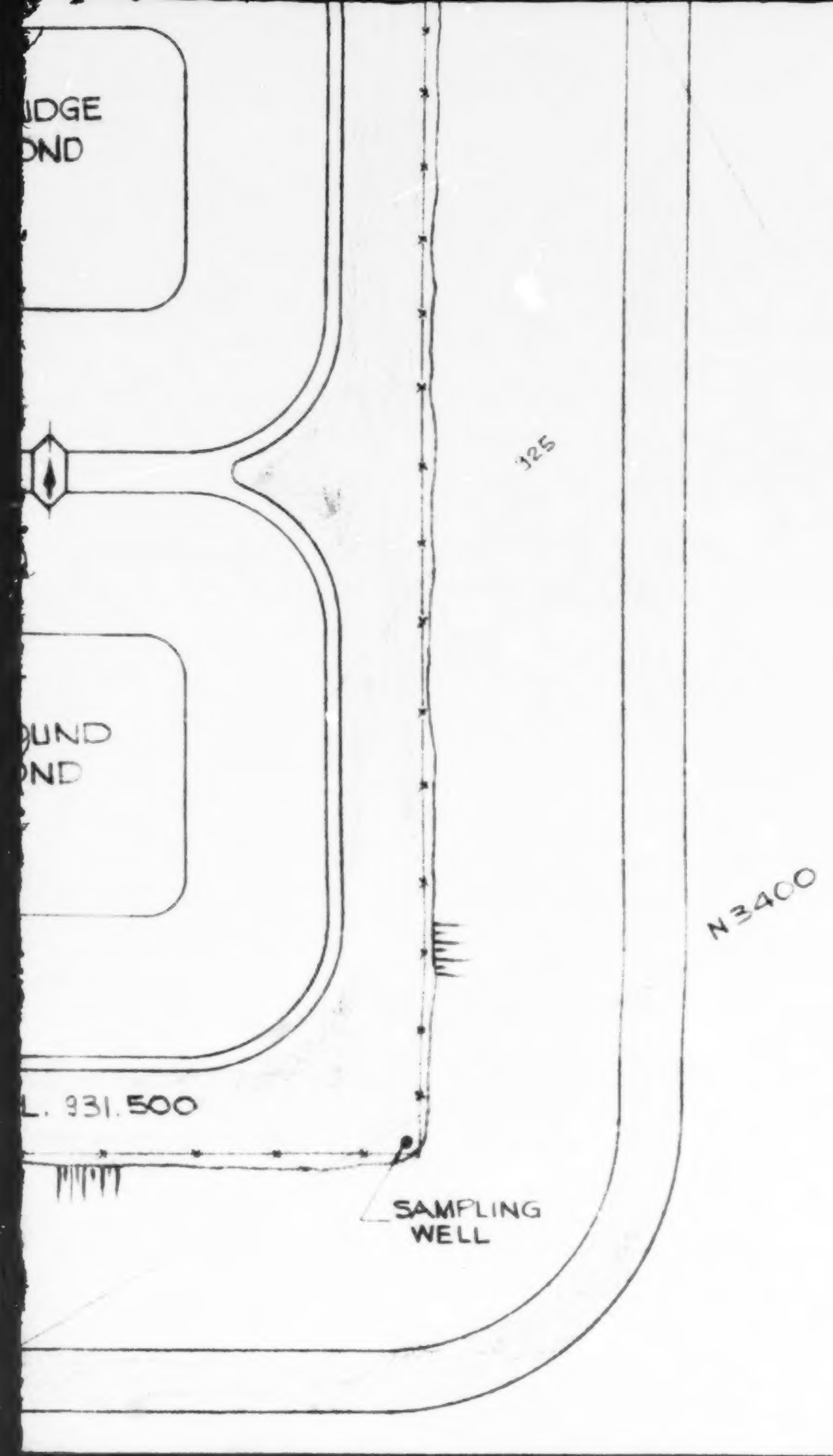
	No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
						P2	MAY 20,87	RELEASED FOR REPORT	J.LI	
						P1		RELEASED FOR INFORMATION	E.N.	

SCALE	1:500	
DRAWN	E.N.	JAN. 7, 1987
CHECKED		
APPROVAL		

WDI

DS

DATE


IDGE
OND
925
UND
OND
N3400
L. 931.500
SAMPLING
WELL

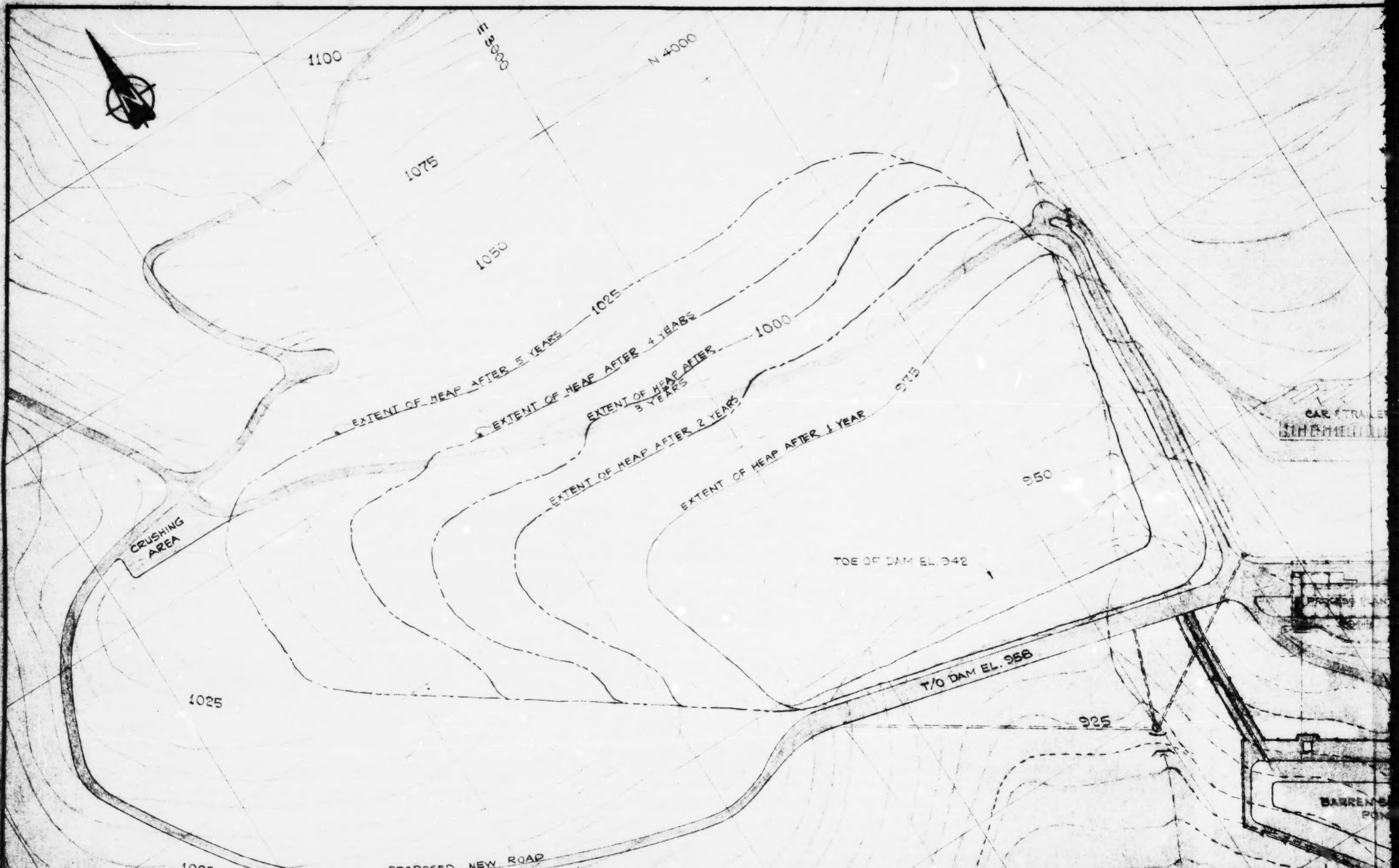
CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEASED FOR REPORT	J.LI	
	P1		RELEASED FOR INFORMATION	E.N.	

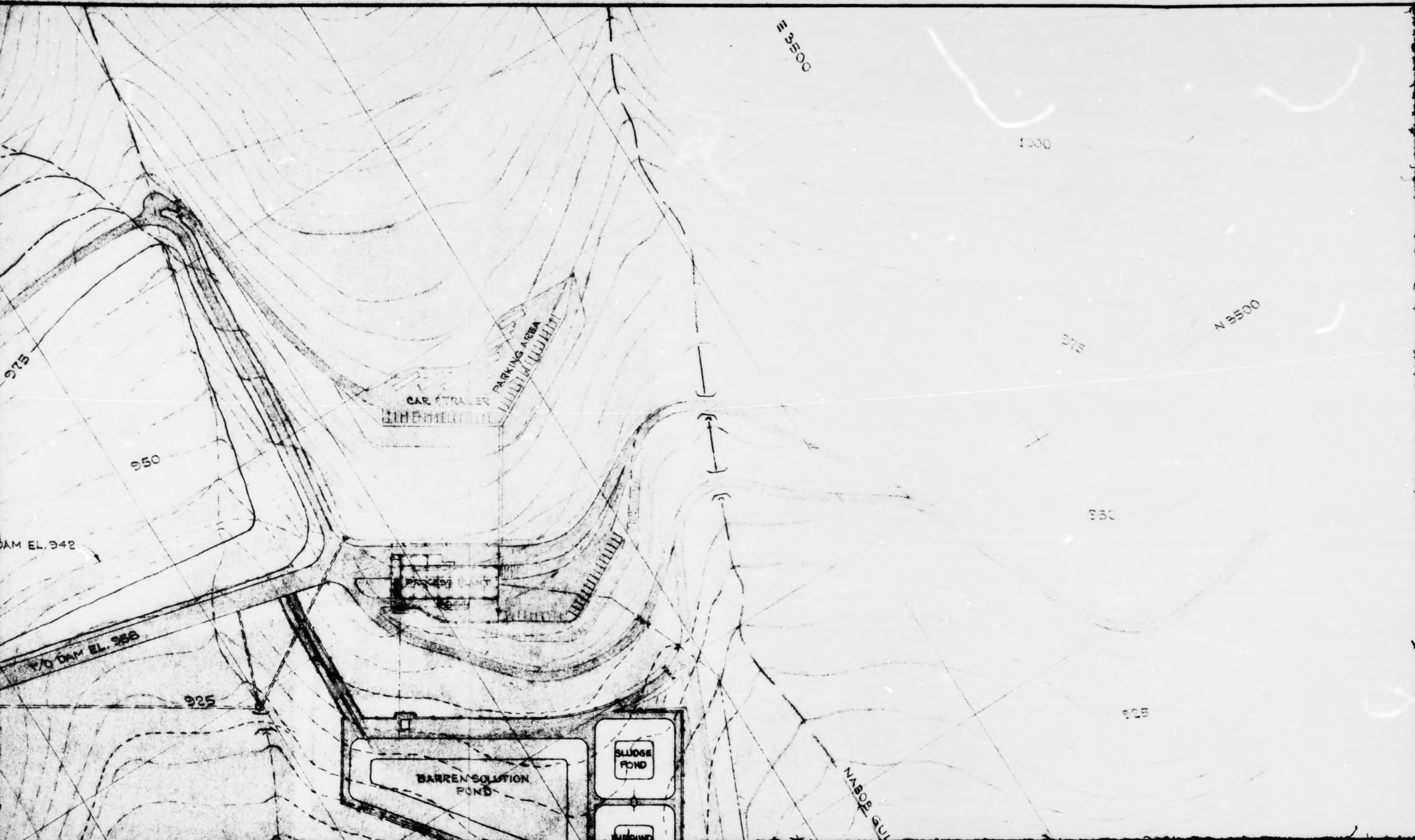
SCALE	1:500	
DRAWN	E.N.	JAN. 7, 1987
CHECKED		
APPROVAL		

WITTECK DEVELOPMENT INC.

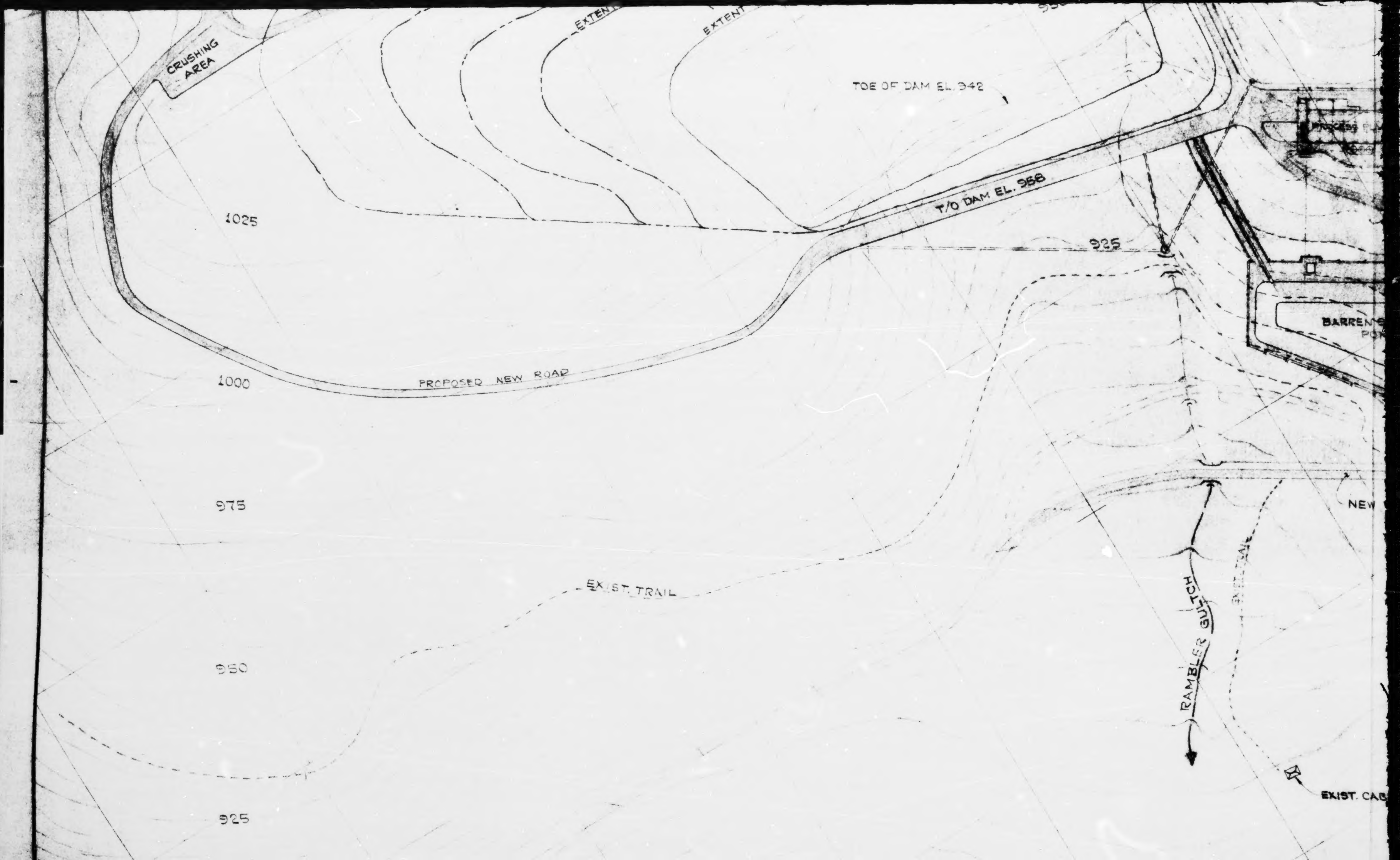
DSS - YUKON HEAP LEACH PROJECT
GENERAL (AREA 10)
LOCAL SITE LAYOUT
PLAN

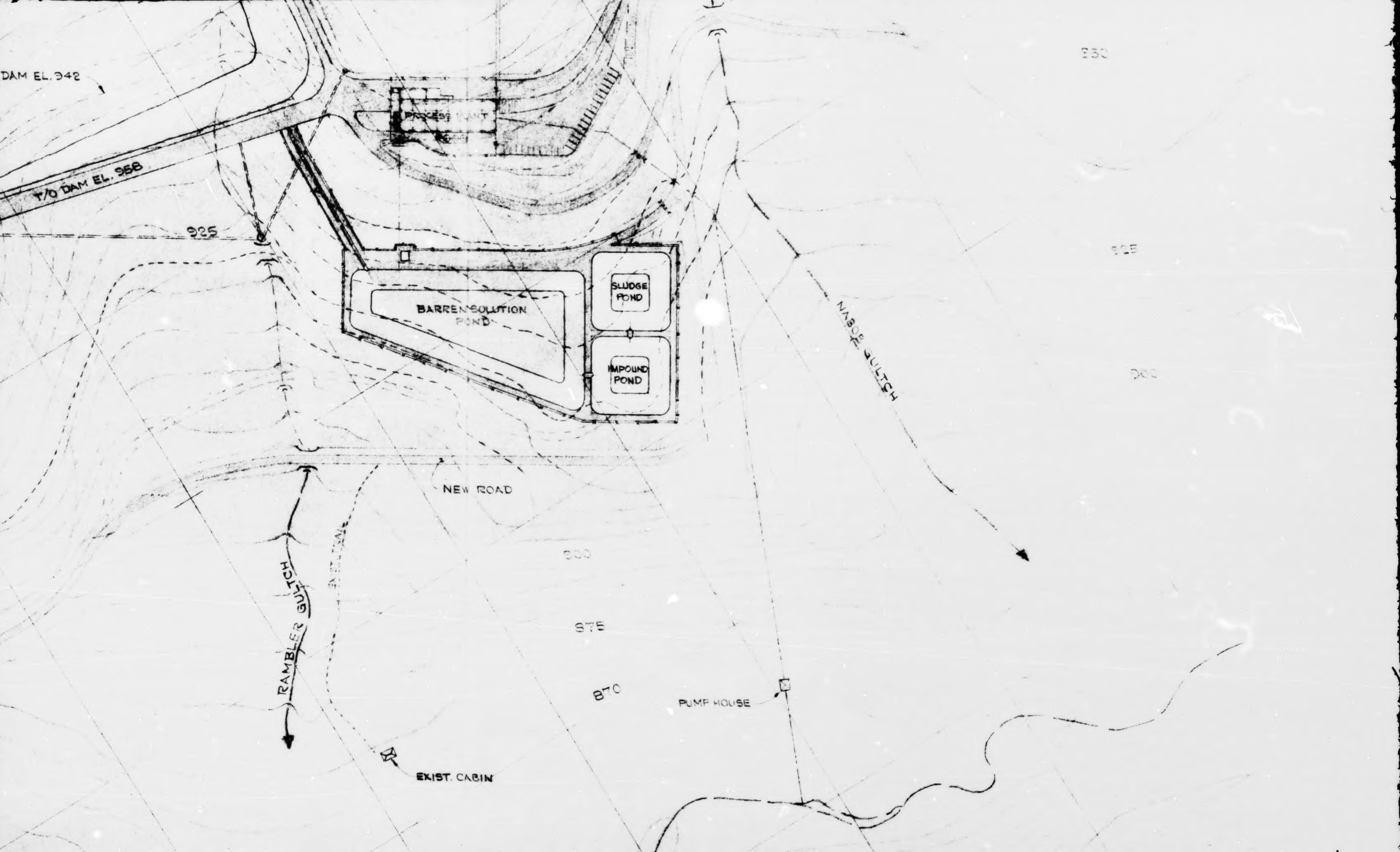
DATE	PROJECT NO	DWG. No	REV.
	5209	10 - 120 - 002	P2

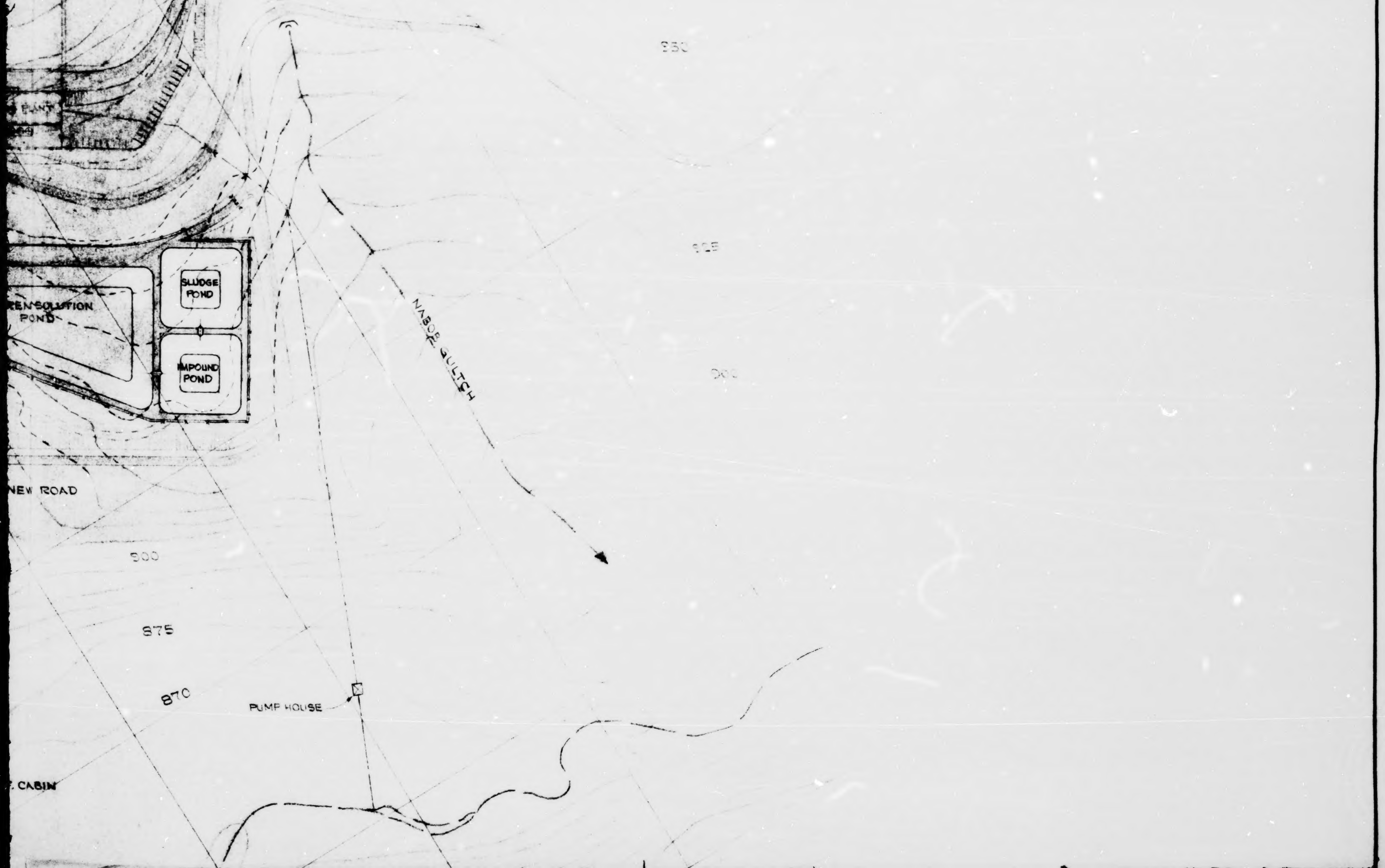

1100
E 3000
N 4000
1075
1050
1025
EXTENT OF HEAP AFTER 5 YEARS
EXTENT OF HEAP AFTER 4 YEARS
1000
EXTENT OF HEAP AFTER 3 YEARS
975
EXTENT OF HEAP AFTER 2 YEARS
EXTENT OF HEAP AFTER 1 YEAR
950
CAR & TRAILE
CRUSHING AREA
TOE OF DAM EL. 942
PROCESS PLAN
T/O DAM EL. 958
1025
925
BARREN S
PON
1000
PROPOSED NEW ROAD


E 3500
1000
N 3500
975
975
PARKING AREA
CAR TRAILER
950
950
DAM EL. 942
PROCESS PLANT
T/O DAM EL. 958
925
925
SLUDGE POND
BARREN SOLUTION POND
NABOB GUL

E 3500
1000
N 3500
975
950
925
PARKING AREA
RAILER
PLANT
SLUDGE POND
NABOB GU

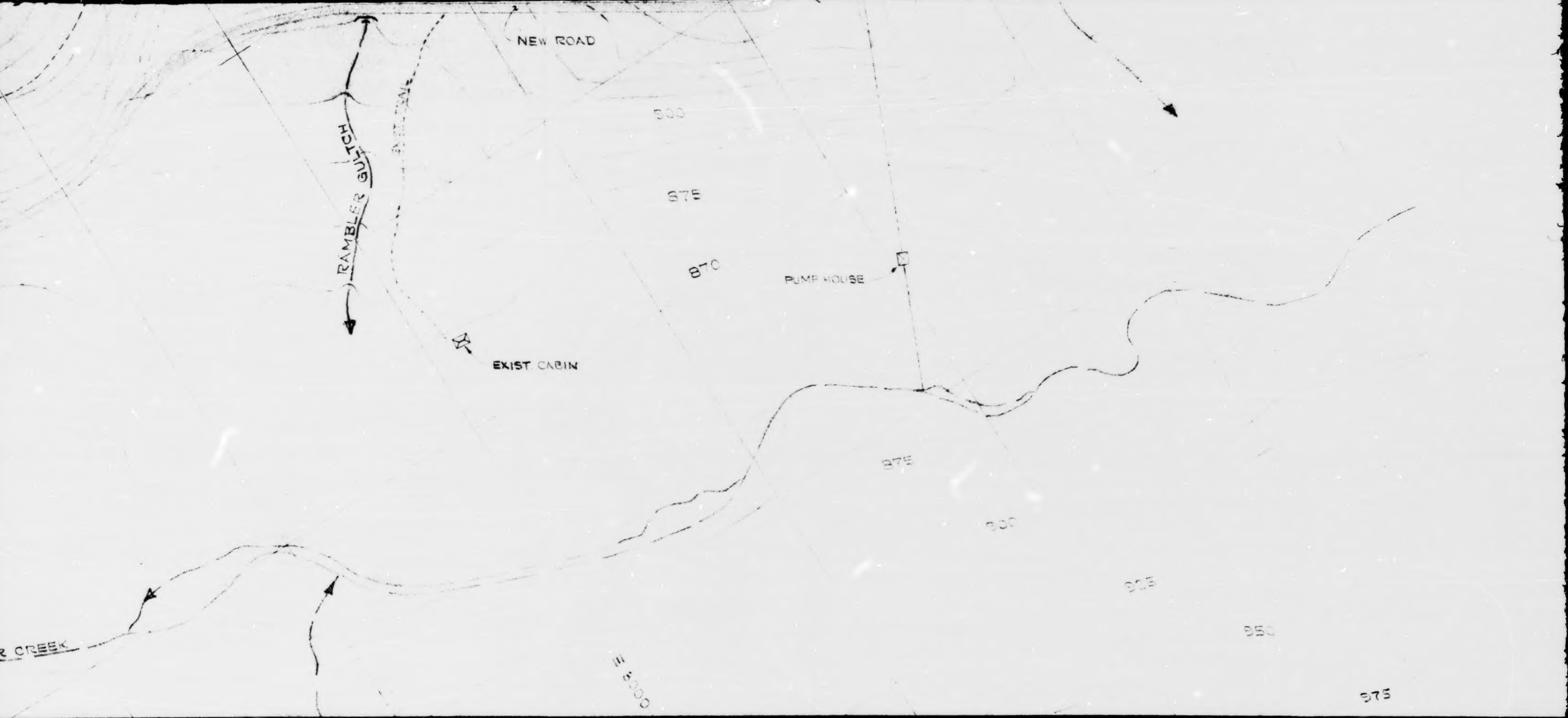

CRUSHING AREA
EXTENT
EXTENT
TOE OF DAM EL. 942
T/O DAM EL. 958
PROCESS P
BARREN
PO
1025
925
1000
PROPOSED NEW ROAD
975
NEW
EXIST. TRAIL
RAMBLER GULTCH
EXIST. TRAIL
950
EXIST. CAB
925


DAM EL. 942
T/O DAM EL. 958
PROCESS PLANT
925
BARREN SOLUTION POND
SLUDGE POND
IMPOUND POND
930
925
900
NABOB GULCH
NEW ROAD
900
EXIST. TRAIL
RAMBLER GULTCH
875
870
PUMP HOUSE
EXIST. CABIN


PLANT
950
925
900
NABOB GULCH
SLUDGE POND
IMPOUND POND
REN SOLUTION POND
NEW ROAD
900
875
870
PUMP HOUSE
CABIN

975
950
925
900
875
EXIST. TRAIL
NEW
RAMBLER GULTCH
EXIST. TRAIL
EXIST. CAB
SEYMOUR CREEK
N3500
E2500

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CHE
40-130-002	PROCESS PLANT LAYOUT - SECTION							
40-130-001	PROCESS PLANT LAYOUT - PLAN							
10-120-002	LOCAL SITE LAYOUT							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND							
30-130-001	HEAP GENERAL LAYOUT							
10-110-002	FLOW DIAGRAM (SHEET 1 OF 2)							
10-110-001	FLOW DIAGRAM (SHEET 2 OF 2)							


NEW ROAD
900
875
870
PUMP HOUSE
RAMBLER GULTCH
EXIST. CABIN
875
900
925
950
975
R CREEK
E 8000

	No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
						P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
						P1		RELEASED FOR INFORMATION	E.N.	

SCALE	1:1500	
DRAWN	C.F.K.	OCT. 1986
CHECKED		
APPROVAL		

WDI

DS

DATE

NEW ROAD
900
875
870
PUMP HOUSE
T. CABIN
875
900
925
950
975
E 5000

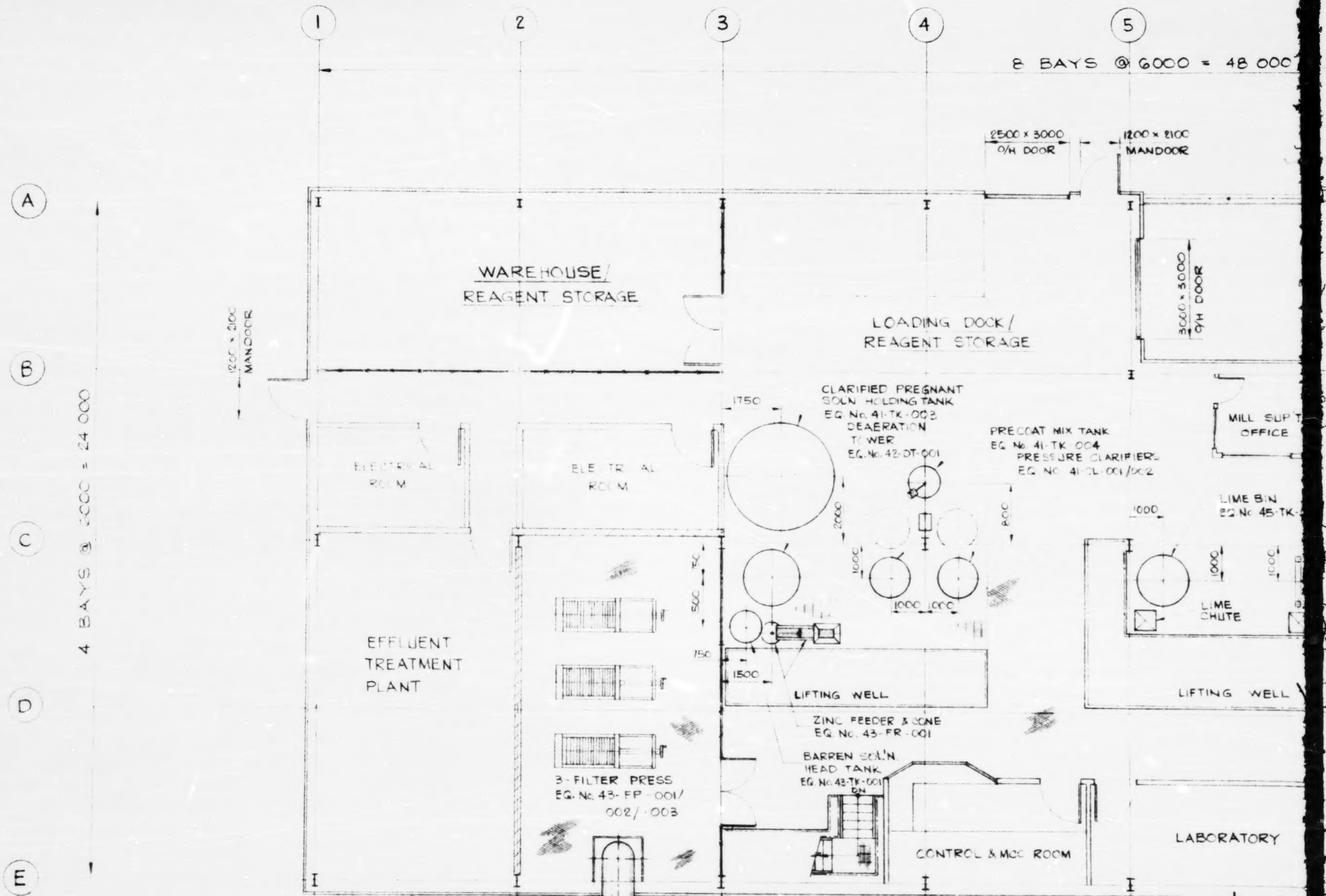
CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
	P1		RELEASED FOR INFORMATION	E.N.	

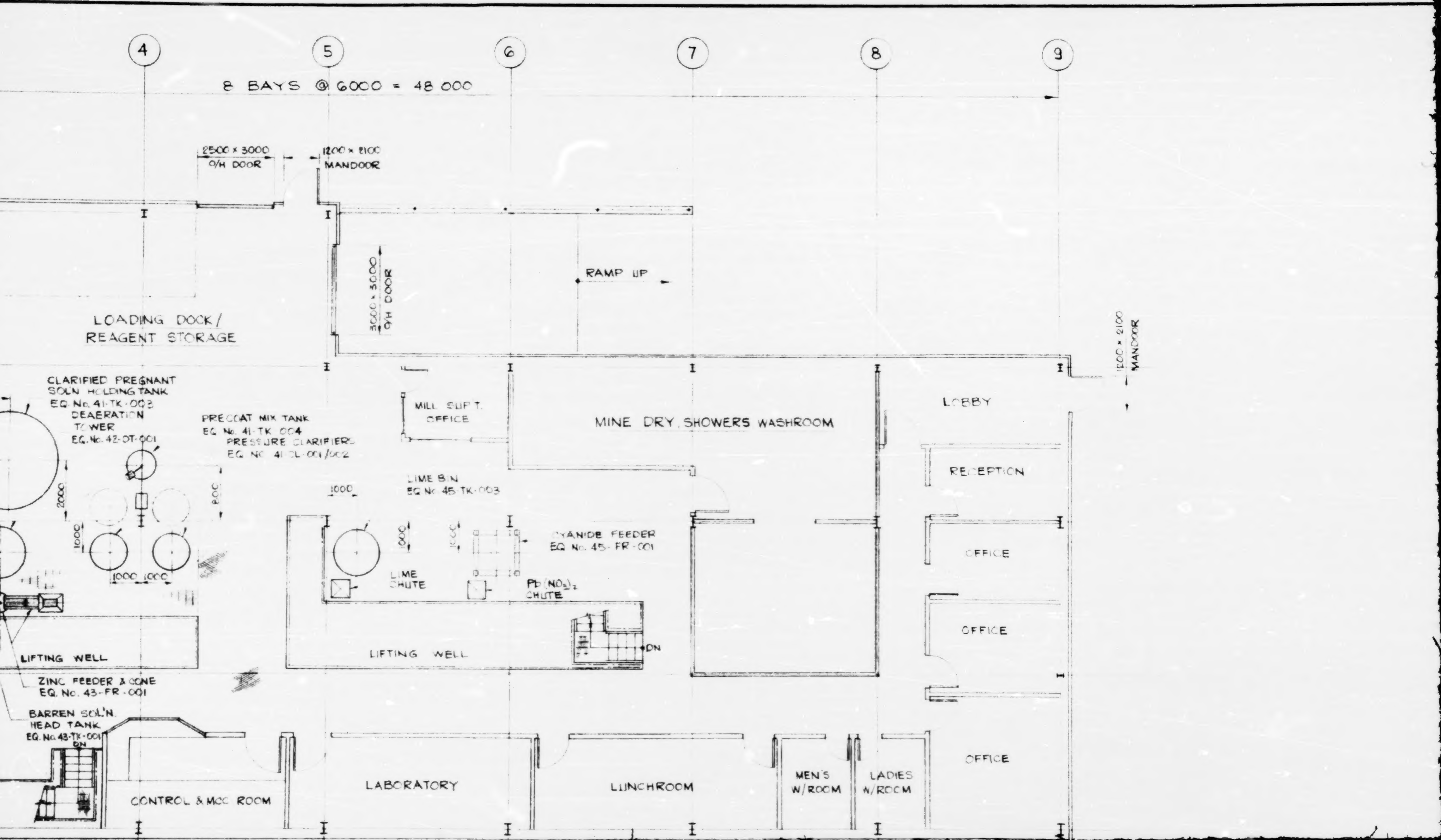
SCALE	1:1500	
DRAWN	C.F.K.	OCT. 1986
CHECKED		
APPROVAL		

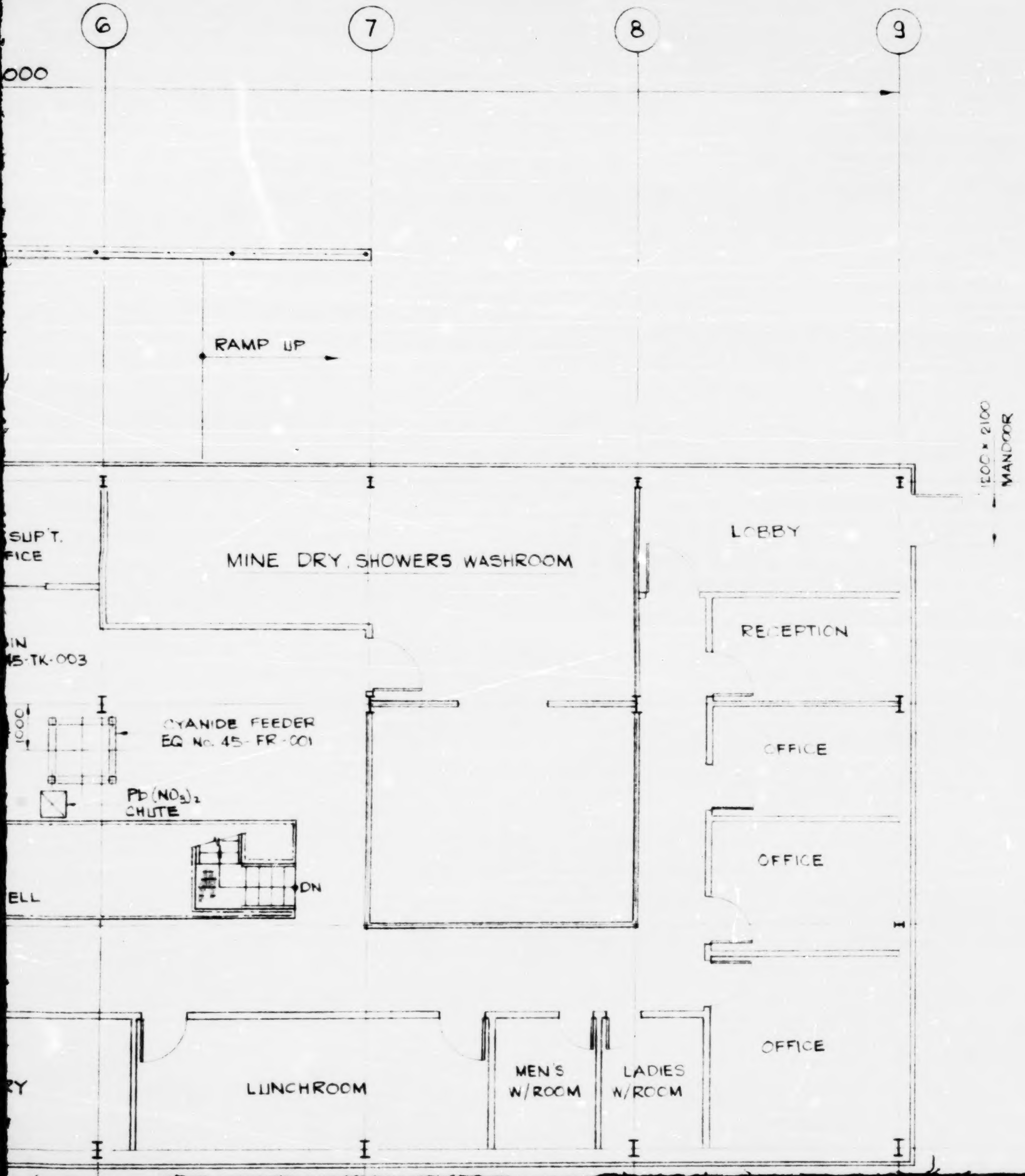
WITTECK DEVELOPMENT INC.

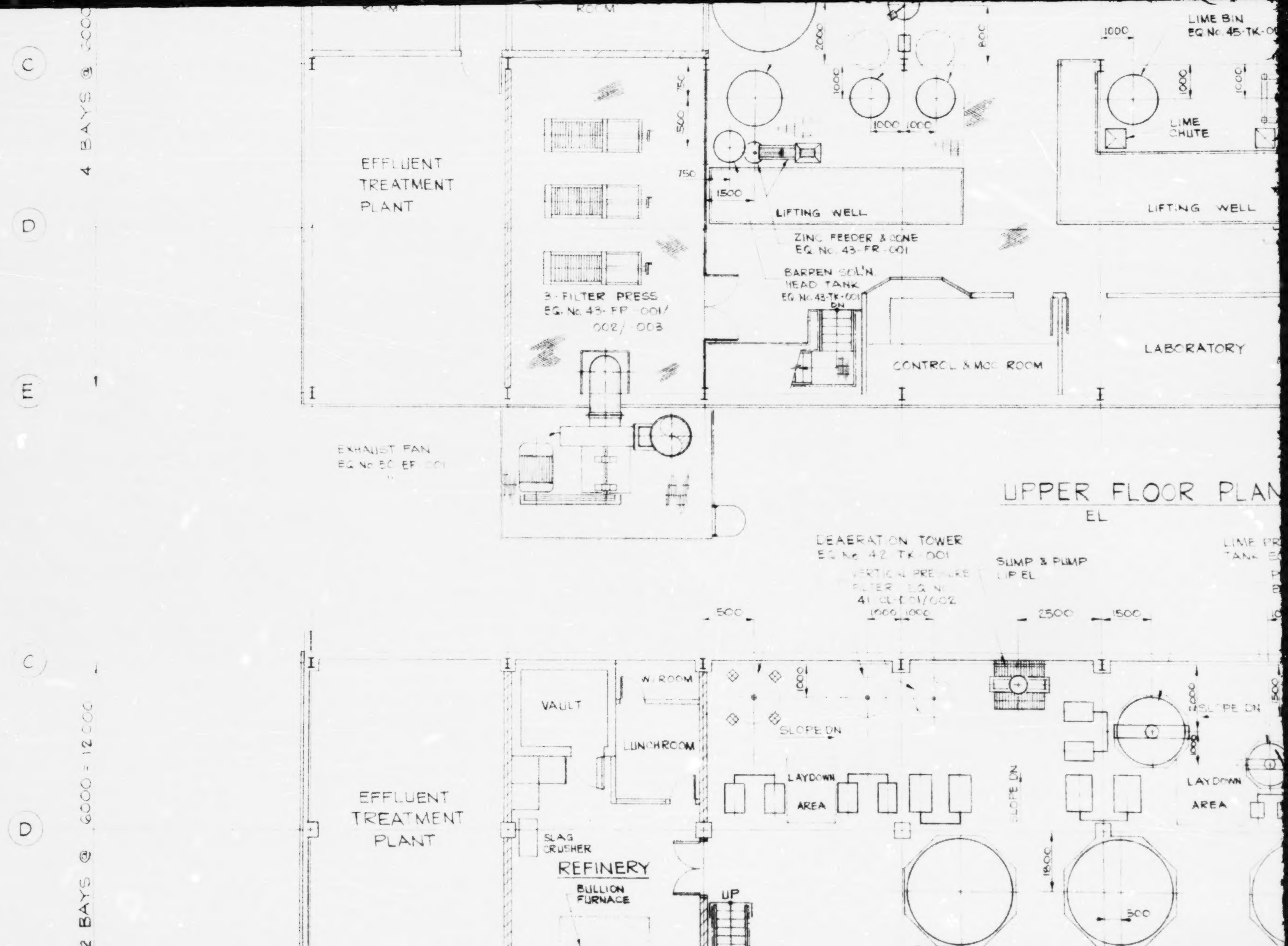
DSS-YUKON HEAP LEACH PROJECT
GENERAL (AREA 10)
GENERAL SITE LAYOUT
PLAN

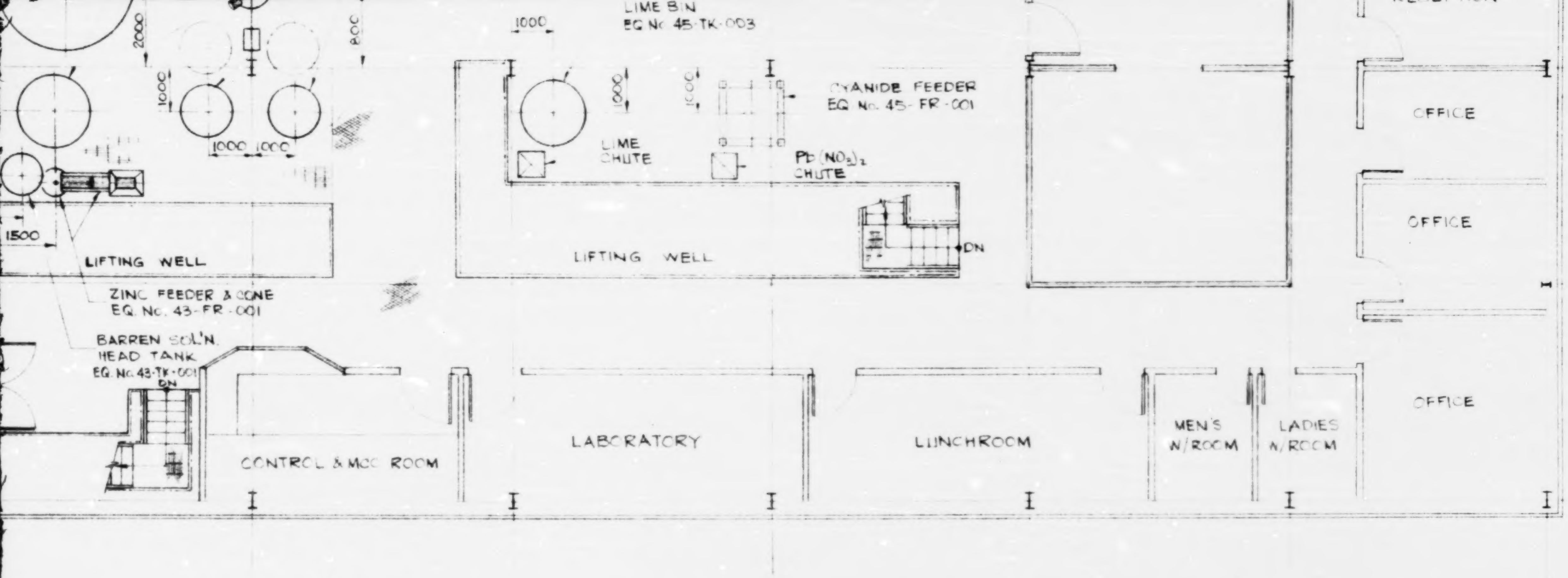
DATE	PROJECT NO.	DWG. No.	REV.
	5209	10-120-001	P2


1
2
3
4
5
8 BAYS @ 6000 = 48 000
A
B
C
D
E
4 BAYS @ 6000 = 24 000
2500 × 3000
O/H DOOR
1200 × 2100
MANDOOR
WAREHOUSE/
REAGENT STORAGE
1200 × 2100
MANDOOR
LOADING DOCK/
REAGENT STORAGE
3000 × 3000
O/H DOOR
ELECTRICAL
ROOM
ELECTRICAL
ROOM
CLARIFIED PREGNANT
SOL'N HOLDING TANK
EQ. No. 41-TK-003
DEAERATION
TOWER
EQ. No. 42-DT-001
PRECOAT MIX TANK
EQ. No. 41-TK-004
PRESSURE CLARIFIERS
EQ. No. 41-CL-001/002
MILL SUP'T
OFFICE
LIME BIN
EQ. No. 45-TK-
1750
2000
800
1000
1000
1000
1000
1000
1000
750
500
150
1500
LIME
CHUTE
EFFLUENT
TREATMENT
PLANT
LIFTING WELL
LIFTING WELL
ZINC FEEDER & CONE
EQ. No. 43-FR-001
BARREN SOL'N.
HEAD TANK
EQ. No. 43-TK-001
DN
3-FILTER PRESS
EQ. No. 43-FP-001/
002/-003
CONTROL & MCC ROOM
LABORATORY


4
5
6
7
8
9
8 BAYS @ 6000 = 48 000
2500 × 3000
O/H DOOR
1200 × 2100
MANDOOR
3000 × 3000
O/H DOOR
RAMP UP
1200 × 2100
MANDOOR
LOADING DOCK/
REAGENT STORAGE
CLARIFIED PREGNANT
SOL'N HOLDING TANK
EQ. No. 41-TK-003
DEAERATION
TOWER
EQ. No. 42-DT-001
PRECOAT MIX TANK
EQ. No. 41-TK-004
PRESSURE CLARIFIERS
EQ. No. 41-CL-001/002
2000
800
1000
1000
1000
MILL SUP'T.
OFFICE
MINE DRY, SHOWERS WASHROOM
LOBBY
RECEPTION
LIME BIN
EQ. No. 45-TK-003
1000
1000
1000
CYANIDE FEEDER
EQ. No. 45-FR-001
LIME
CHUTE
Pb(NO3)2
CHUTE
OFFICE
OFFICE
DN
LIFTING WELL
LIFTING WELL
ZINC FEEDER & CONE
EQ. No. 43-FR-001
BARREN SOL'N.
HEAD TANK
EQ. No. 43-TK-001
DN
CONTROL & MCC ROOM
LABORATORY
LUNCHROOM
MEN'S
W/ROOM
LADIES
W/ROOM
OFFICE

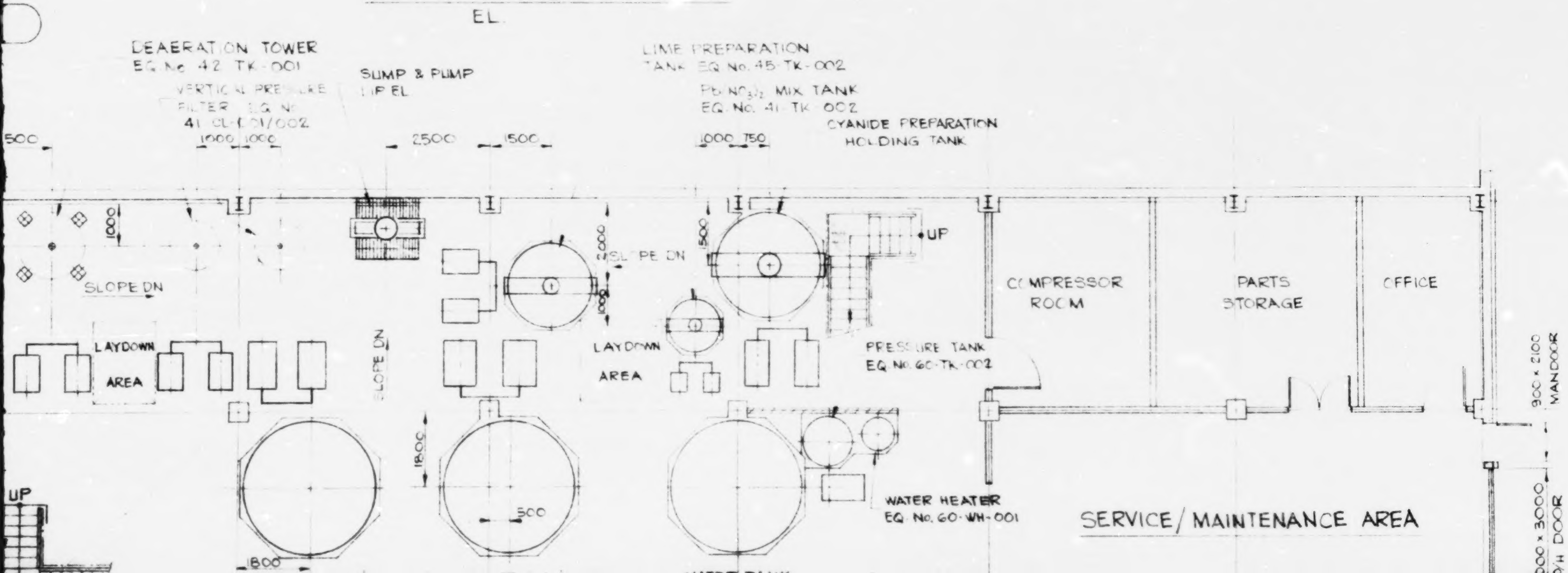

6
7
8
9
000
RAMP UP
SUP'T.
FICE
MINE DRY, SHOWERS WASHROOM
LOBBY
1200 × 2100
MANDOOR
RECEPTION
OFFICE
OFFICE
OFFICE
IN
45-TK-003
1000
CYANIDE FEEDER
EQ No. 45-FR-001
Pb(NO3)2
CHUTE
DN
ELL
RY
LUNCHROOM
MEN'S
W/ROOM
LADIES
W/ROOM

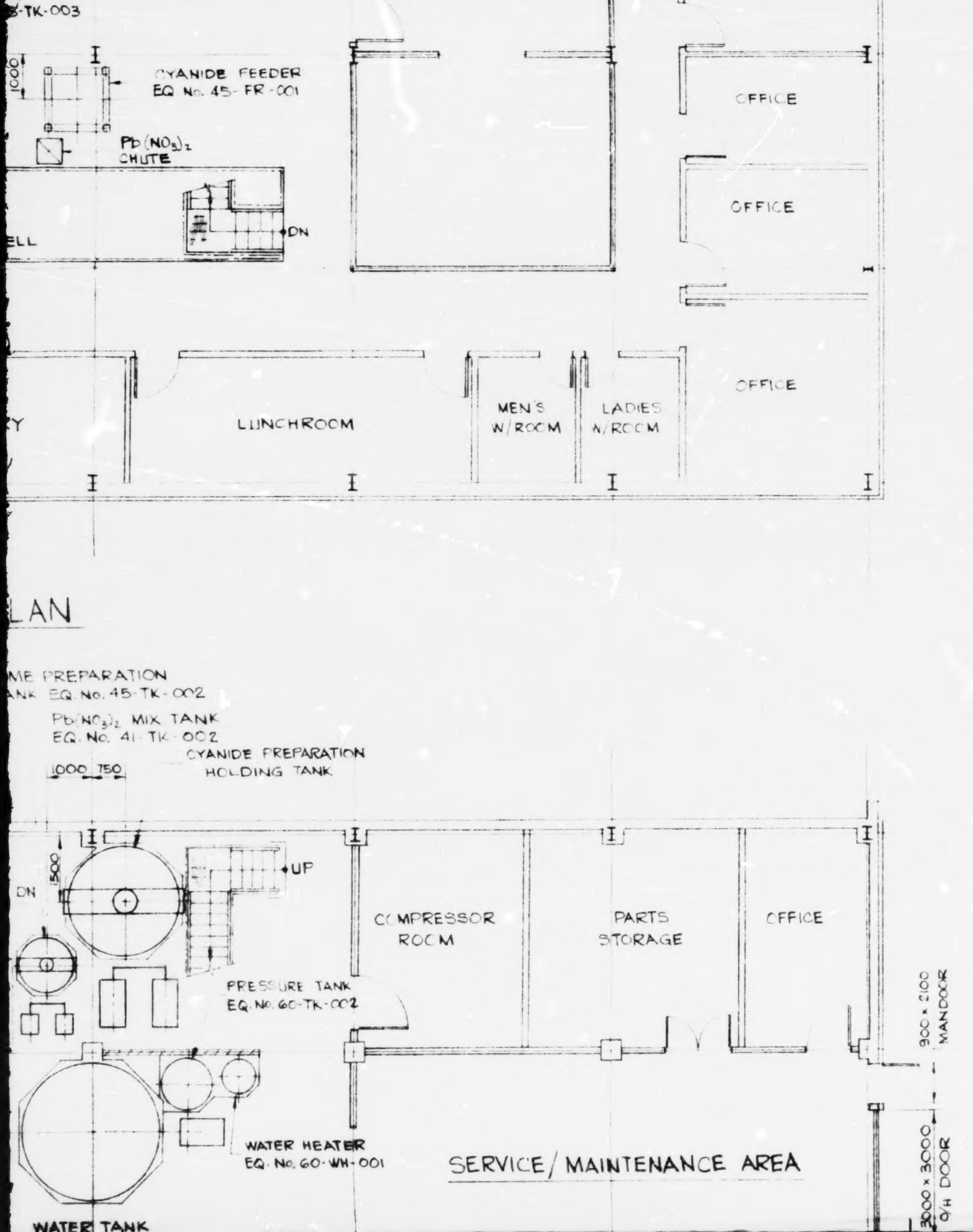

ROOM
ROOM
LIME BIN
EQ No. 45-TK-0
1000
2000
800
1000
1000 1000
1000
1000
C
D
E
4 BAYS @ 3000
750
500
750
1500
EFFLUENT
TREATMENT
PLANT
LIME
CHUTE
LIFTING WELL
LIFTING WELL
ZINC FEEDER & CONE
EQ. No. 43-FR-001
BARREN SOL'N.
HEAD TANK
EQ No. 43-TK-001
DN
3-FILTER PRESS
EQ. No. 43-FP-001/
002/-003
CONTROL & MCC ROOM
LABORATORY
EXHAUST FAN
EQ No. 50-EF-001
UPPER FLOOR PLAN
EL
DEAERATION TOWER
EQ No. 42-TK-001
VERTICAL PRESSURE
FILTER EQ. No.
41-CL-001/002
1000 1000
SUMP & PUMP
UP EL
LIME PR
TANK E
500
2500
1500
C
D
2 BAYS @ 6000 = 12000
W/ROOM
VAULT
LUNCHROOM
SLOPE DN
1000
2000
SLOPE DN
1000
LAYDOWN
AREA
SLOPE DN
LAYDOWN
AREA
500
EFFLUENT
TREATMENT
PLANT
SLAG
CRUSHER
REFINERY
BULLION
FURNACE
UP
1800
500

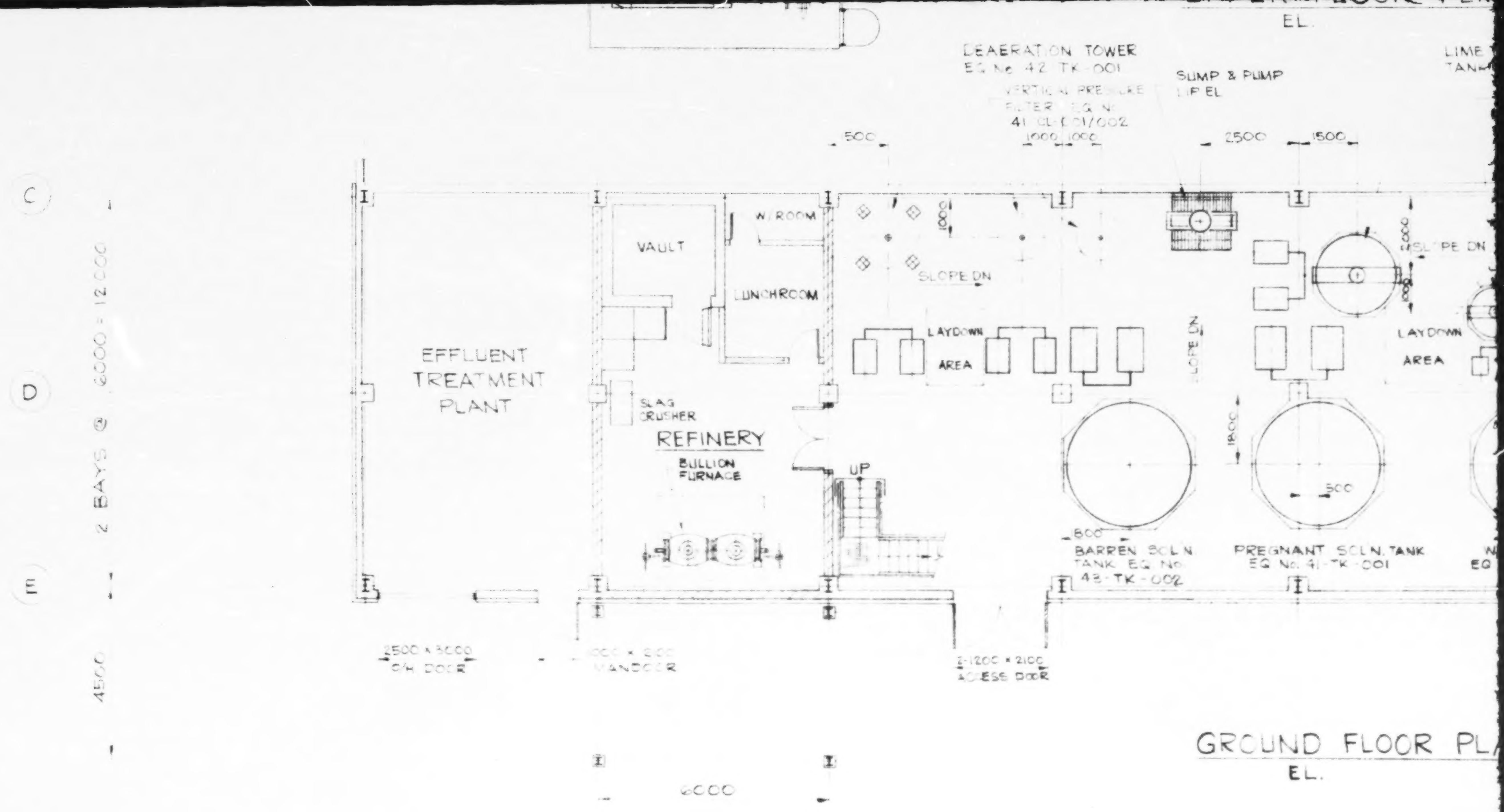

LIME BIN
EQ No 45-TK-003
1000
2000
800
1000
1000
1000 1000
CYANIDE FEEDER
EQ No. 45-FR-001
LIME
CHUTE
$Pb(NO_3)_2$
CHUTE
OFFICE
OFFICE
1500
LIFTING WELL
LIFTING WELL
DN
ZINC FEEDER & CONE
EQ. No. 43-FR-001
BARREN SOL'N.
HEAD TANK
EQ. No. 43-TK-001
DN
OFFICE
CONTROL & MCC ROOM
LABORATORY
LUNCHROOM
MEN'S
W/ROOM
LADIES
W/ROOM

UPPER FLOOR PLAN

EL.


DEAERATION TOWER
EQ No. 42 TK-001
VERTICAL PRESSURE
FILTER EQ No
41 CL-001/002
SUMP & PUMP
UP EL.
LIME PREPARATION
TANK EQ No. 45-TK-002
$Pb(NO_3)_2$ MIX TANK
EQ. No. 41-TK-002
CYANIDE PREPARATION
HOLDING TANK
500
1000 1000
2500
1500
1000 750
1000
SLOPE DN
1000
SLOPE DN
1000
500
UP
SLOPE DN
LAYDOWN
AREA
LAYDOWN
AREA
COMPRESSOR
ROOM
PARTS
STORAGE
OFFICE
PRESSURE TANK
EQ. No. 60-TK-002
900 × 2100
MANDOOR
1800
500
UP
1800
WATER HEATER
EQ. No. 60-WH-001
SERVICE/MAINTENANCE AREA
000 × 3000
O/H DOOR


-TK-003
1000
CYANIDE FEEDER
EQ No. 45-FR-001
Pb(NO3)2
CHUTE
ELL
DN
OFFICE
OFFICE
OFFICE
LUNCHROOM
MEN'S
W/ROOM
LADIES
W/ROOM
Y
LAN
ME PREPARATION
ANK EQ. No. 45-TK-002
Pb(NO3)2 MIX TANK
EQ. No. 41-TK-002
CYANIDE PREPARATION
HOLDING TANK
1000
750
1500
DN
UP
COMPRESSOR
ROOM
PARTS
STORAGE
OFFICE
PRESSURE TANK
EQ. No. 60-TK-002
900 × 2100
MANDOOR
WATER HEATER
EQ. No. 60-WH-001
SERVICE/MAINTENANCE AREA
3000 × 3000
O/H DOOR
WATER TANK

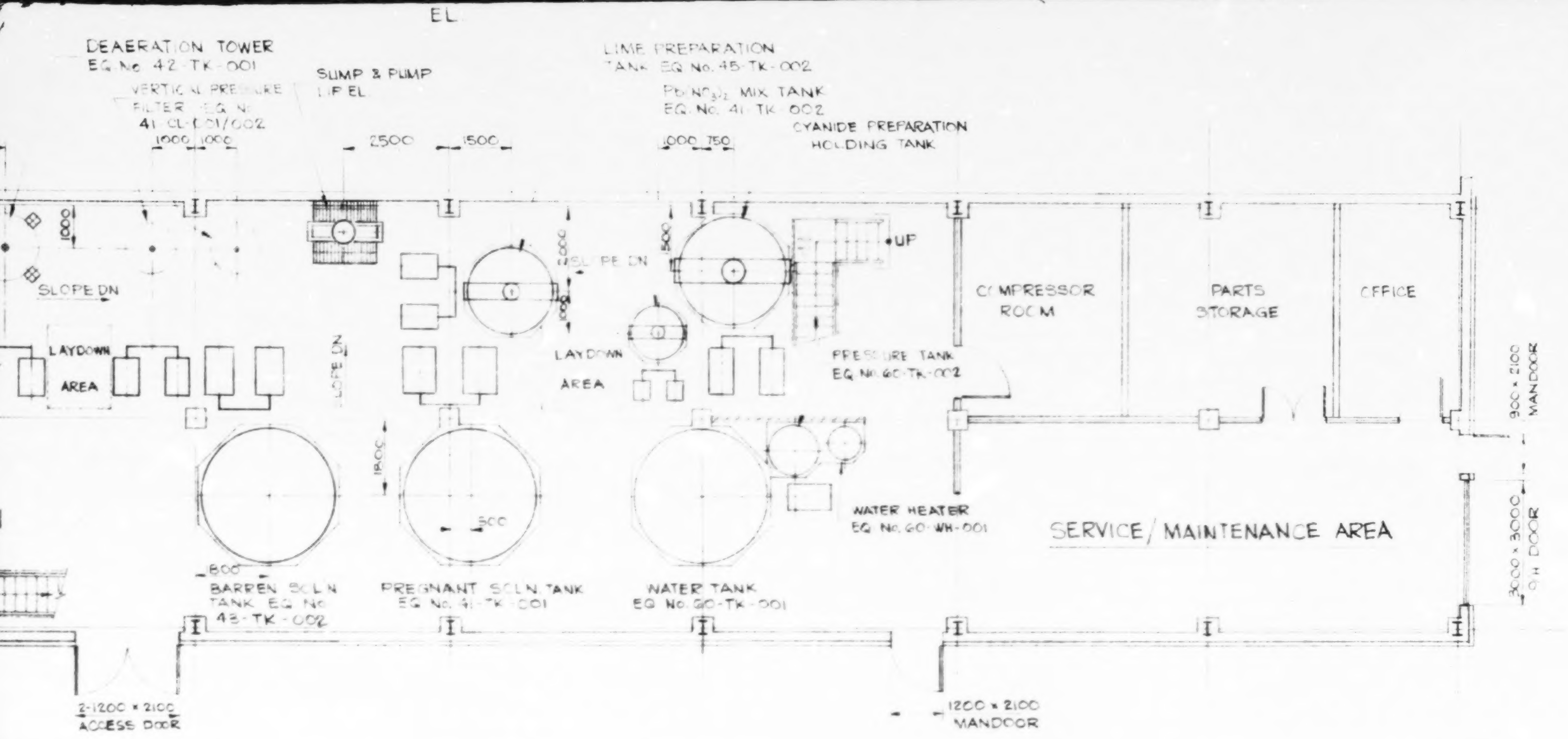

EL.
DEAERATION TOWER
EQ No. 42-TK-001
VERTICAL PRESSURE FILTER EQ No. 41-CL-001/002
SUMP & PUMP
LIME TANK
500
1000 1000
2500
1500
C
D
E
2 BAYS @ 6000 = 12000
4500
W/ROOM
VAULT
LUNCHROOM
SLOPE DN
LAYDOWN AREA
EFFLUENT TREATMENT PLANT
SLAG CRUSHER
REFINERY
BULLION FURNACE
UP
1800
500
800
BARREN SOL'N TANK EQ No. 43-TK-002
PREGNANT SOL'N TANK EQ No. 41-TK-001
EQ
2500 x 3000 O/H DOOR
1000 x 2100 MANDOOR
2-1200 x 2100 ACCESS DOOR
6000

GROUND FLOOR PL

EL.

DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	C
40-130-002	PROCESS PLANT LAYOUT - SECTION							
30-130-003	IMPOUND POND GENERAL LAYOUT							
30-130-002	BARREN SOLUTION POND GENERAL LAYOUT							
30-130-001	HEAP GENERAL LAYOUT							
10-120-002	LOCAL SITE LAYOUT							
10-120-001	GENENRAL SITE LAYOUT							
10-110-002	FLOW DIAGRAM (SHEET 2 OF 2)							
10-110-001	FLOW DIAGRAM (SHEET 1 OF 2)							

TRANSTEX


EL.
DEAERATION TOWER
EQ. No. 42-TK-001
VERTICAL PRESSURE
FILTER EQ. No.
41-CL-001/002
SUMP & PUMP
UP EL.
LIME PREPARATION
TANK EQ. No. 45-TK-002
Pb(NO3)2 MIX TANK
EQ. No. 41-TK-002
CYANIDE PREPARATION
HOLDING TANK
1000
1000
2500
1500
1000
750
1000
SLOPE DN
LAYDOWN
AREA
SLOPE DN
2000
SLOPE DN
1000
500
UP
LAYDOWN
AREA
COMPRESSOR
ROOM
PARTS
STORAGE
OFFICE
PRESSURE TANK
EQ. No. 60-TK-002
900 × 2100
MANDOOR
1800
500
WATER HEATER
EQ. No. 60-WH-001
SERVICE/MAINTENANCE AREA
3000 × 3000
O/H DOOR
800
BARREN SOL'N
TANK EQ. No.
43-TK-002
PREGNANT SOL'N TANK
EQ. No. 41-TK-001
WATER TANK
EQ. No. 60-TK-001
2-1200 × 2100
ACCESS DOOR
1200 × 2100
MANDOOR

GROUND FLOOR PLAN
EL.

	No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK	SCALE	1:100	
											DRAWN	E.N	13 JAN.1987
											CHECKED		
											APPROVAL		
						P2	MAY 20,87	RELEASED FOR REPORT	J. LI				
						P1		RELEAED FOR INFORMATION	E.N.				

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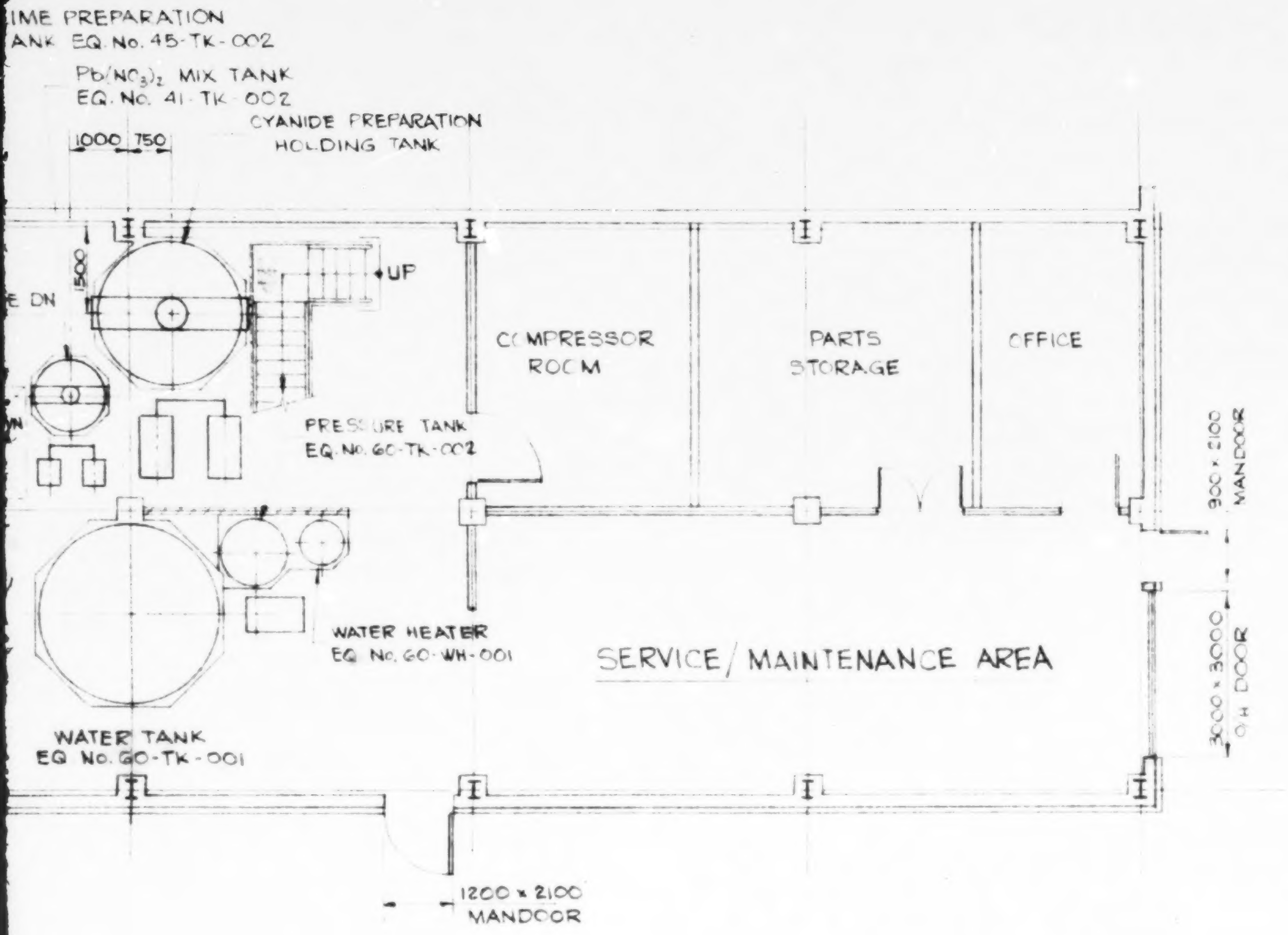

IME PREPARATION
ANK EQ. No. 45-TK-002
$Pb(NO_3)_2$ MIX TANK
EQ. No. 41-TK-002
CYANIDE PREPARATION
HOLDING TANK
1000
750
1500
UP
E DN
COMPRESSOR
ROOM
PARTS
STORAGE
OFFICE
PRESSURE TANK
EQ. No. 60-TK-002
900 × 2100
MANDOOR
WATER HEATER
EQ. No. 60-WH-001
SERVICE/MAINTENANCE AREA
3000 × 3000
O.H. DOOR
WATER TANK
EQ. No. 60-TK-001
1200 × 2100
MANDOOR

PLAN

CHECK	No.	DATE	REVISIONS	BY	CHECK
	P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
	P1		RELEAED FOR INFORMATION	E.N.	

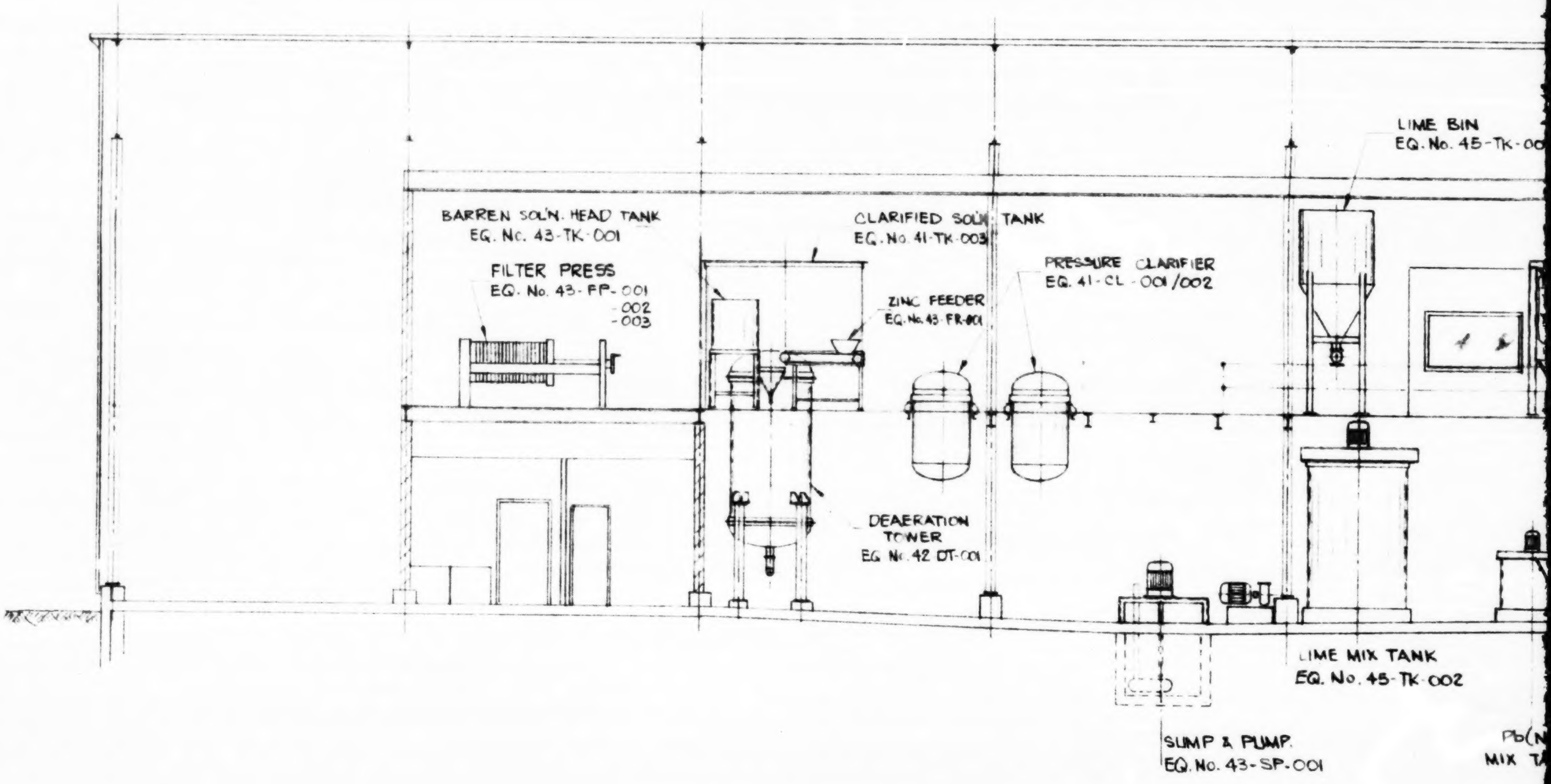
SCALE	1:100	
DRAWN	E. N	13 JAN. 1987
CHECKED		
APPROVAL		

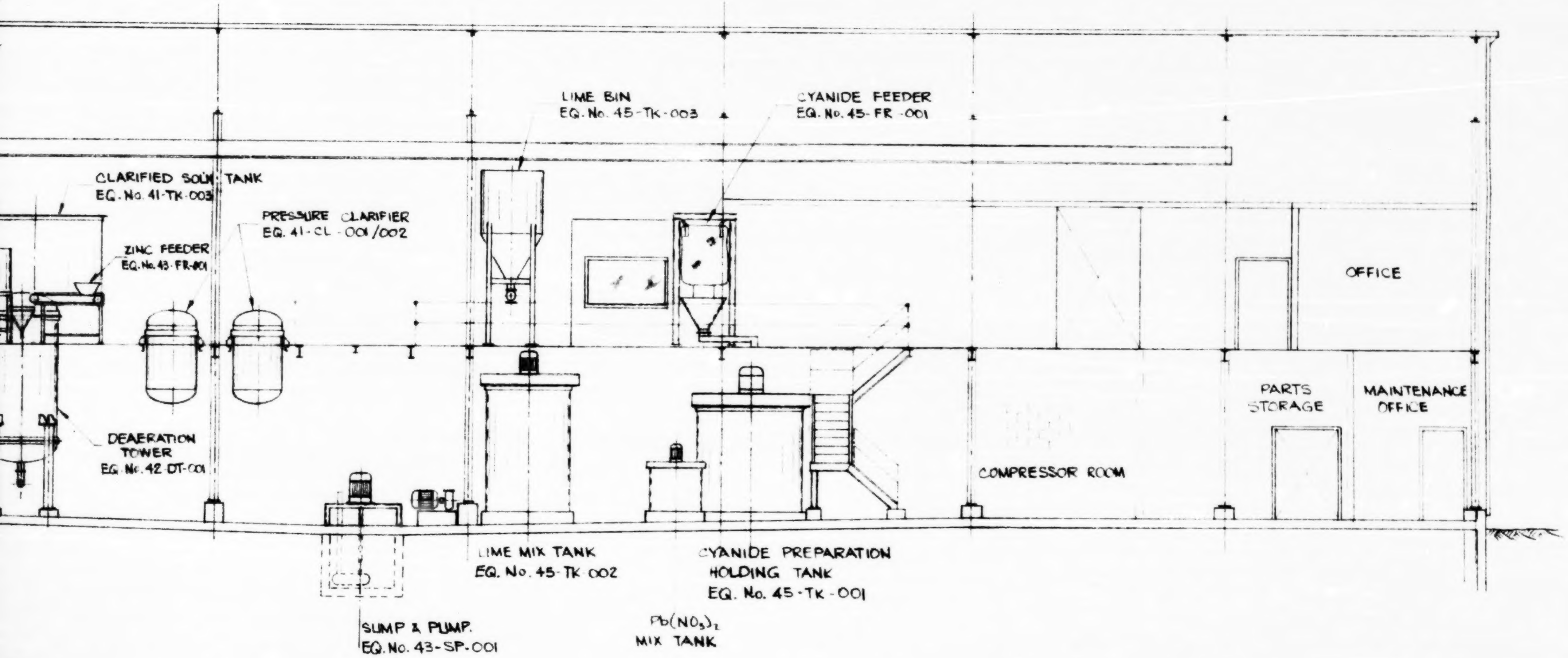
WITTECK DEVELOPMENT INC.

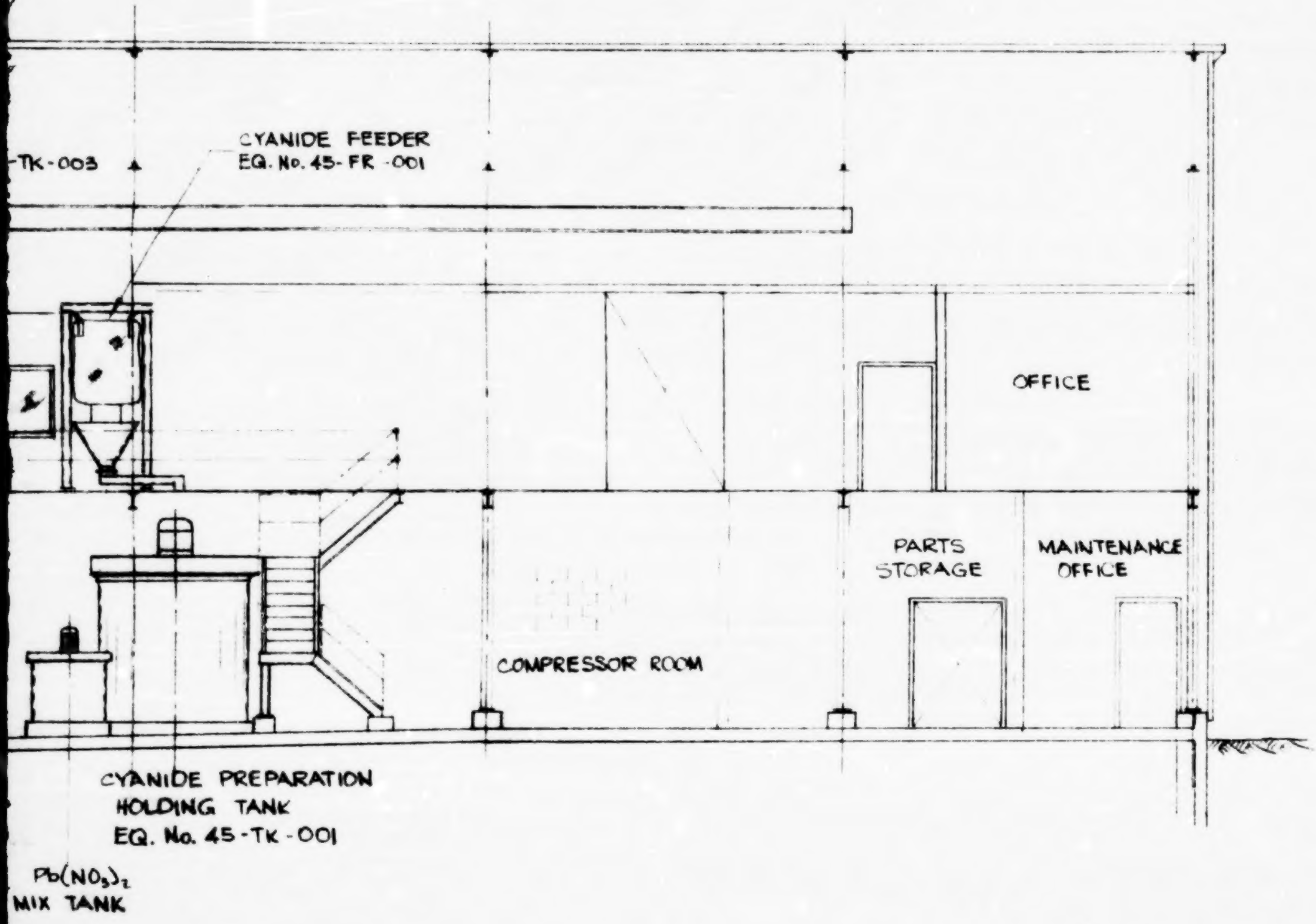
DSS-YUKON HEAP LEACH PROJECT
PROCESS PLANT (AREA 40)
PROCESS PLANT LAYOUT
PLAN

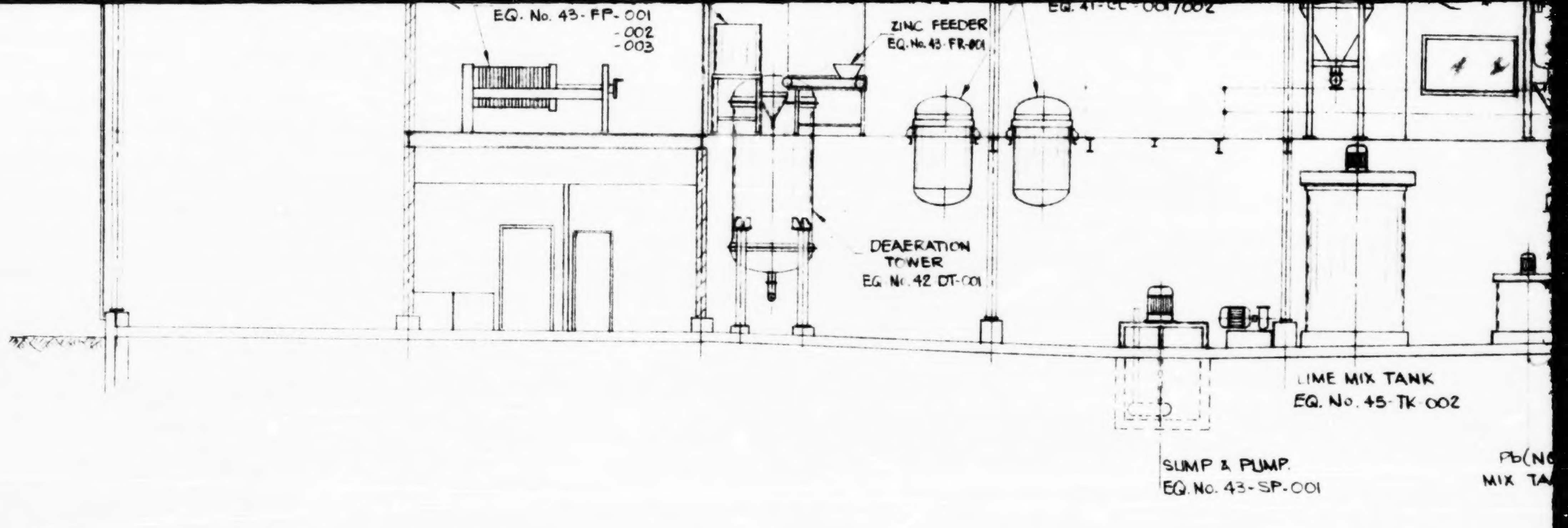
DATE	PROJECT NO.	DWG. No.	REV.
	5209	40-130-001	P2

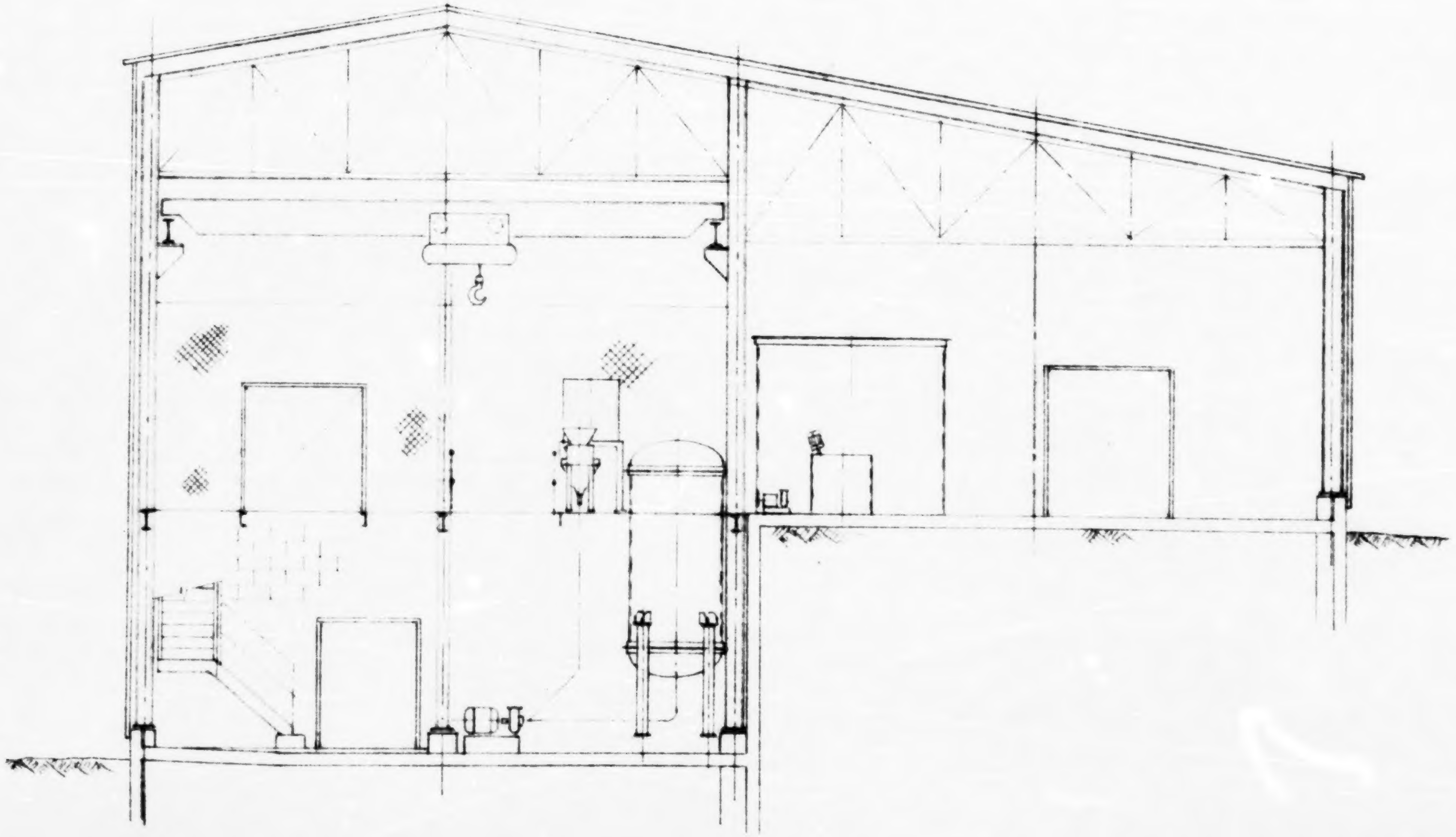
1

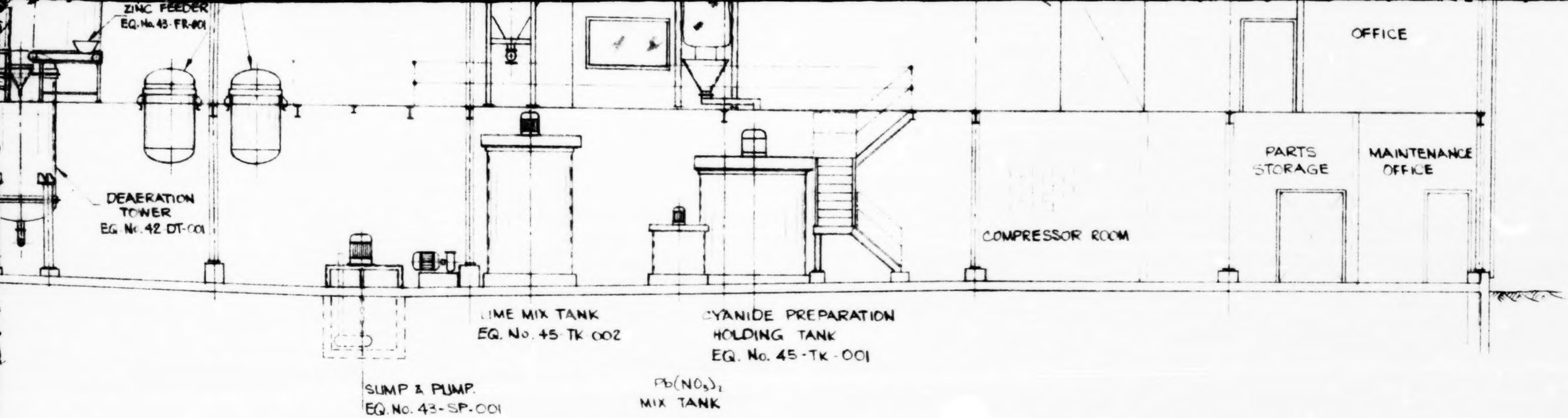

LIME BIN
EQ. No. 45-TK-00
BARREN SOL'N. HEAD TANK
EQ. No. 43-TK-001
CLARIFIED SOL'N. TANK
EQ. No. 41-TK-003
FILTER PRESS
EQ. No. 43-FP-001
-002
-003
ZINC FEEDER
EQ. No. 43-FR-001
PRESSURE CLARIFIER
EQ. 41-CL-001/002
DEAERATION
TOWER
EQ. No. 42 DT-001
LIME MIX TANK
EQ. No. 45-TK-002
SUMP & PUMP.
EQ. No. 43-SP-001
Pb(N
MIX TA

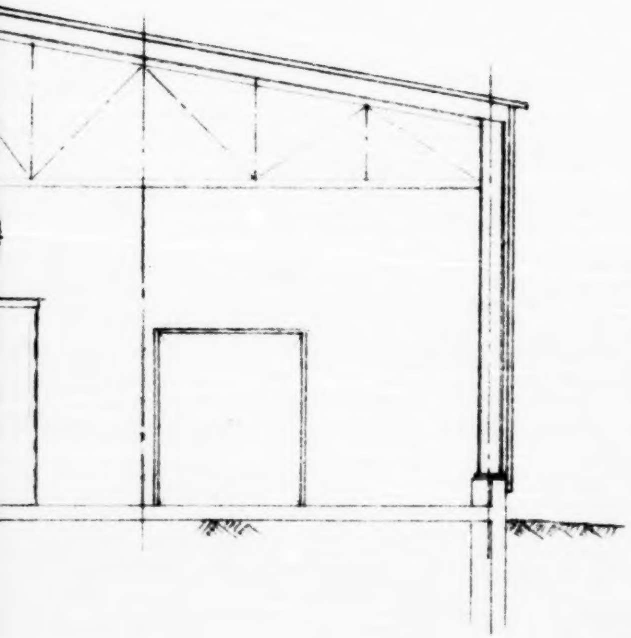

CLARIFIED SOLN. TANK
EQ. No. 41-TK-003
PRESSURE CLARIFIER
EQ. 41-CL-001/002
ZINC FEEDER
EQ. No. 43-FR-001
LIME BIN
EQ. No. 45-TK-003
CYANIDE FEEDER
EQ. No. 45-FR-001
OFFICE
DEAERATION
TOWER
EQ. No. 42 DT-001
PARTS
STORAGE
MAINTENANCE
OFFICE
COMPRESSOR ROOM
LIME MIX TANK
EQ. No. 45-TK 002
CYANIDE PREPARATION
HOLDING TANK
EQ. No. 45-TK-001
SUMP & PUMP.
EQ. No. 43-SP-001
Pb(NO3)2
MIX TANK

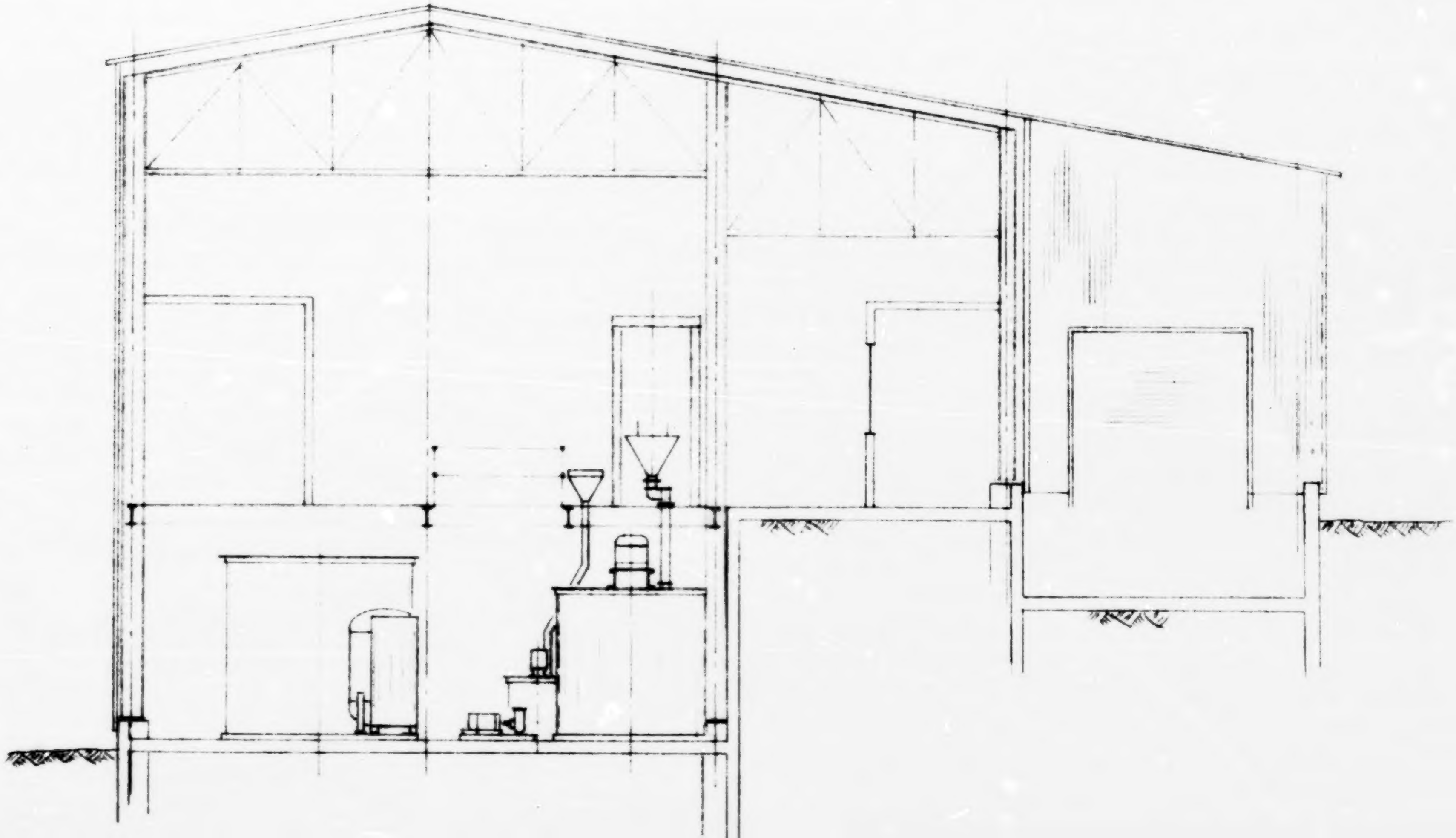

-TK-003
CYANIDE FEEDER
EQ. No. 45-FR-001
OFFICE
PARTS
STORAGE
MAINTENANCE
OFFICE
COMPRESSOR ROOM
CYANIDE PREPARATION
HOLDING TANK
EQ. No. 45-TK-001
Pb(NO3)2
MIX TANK

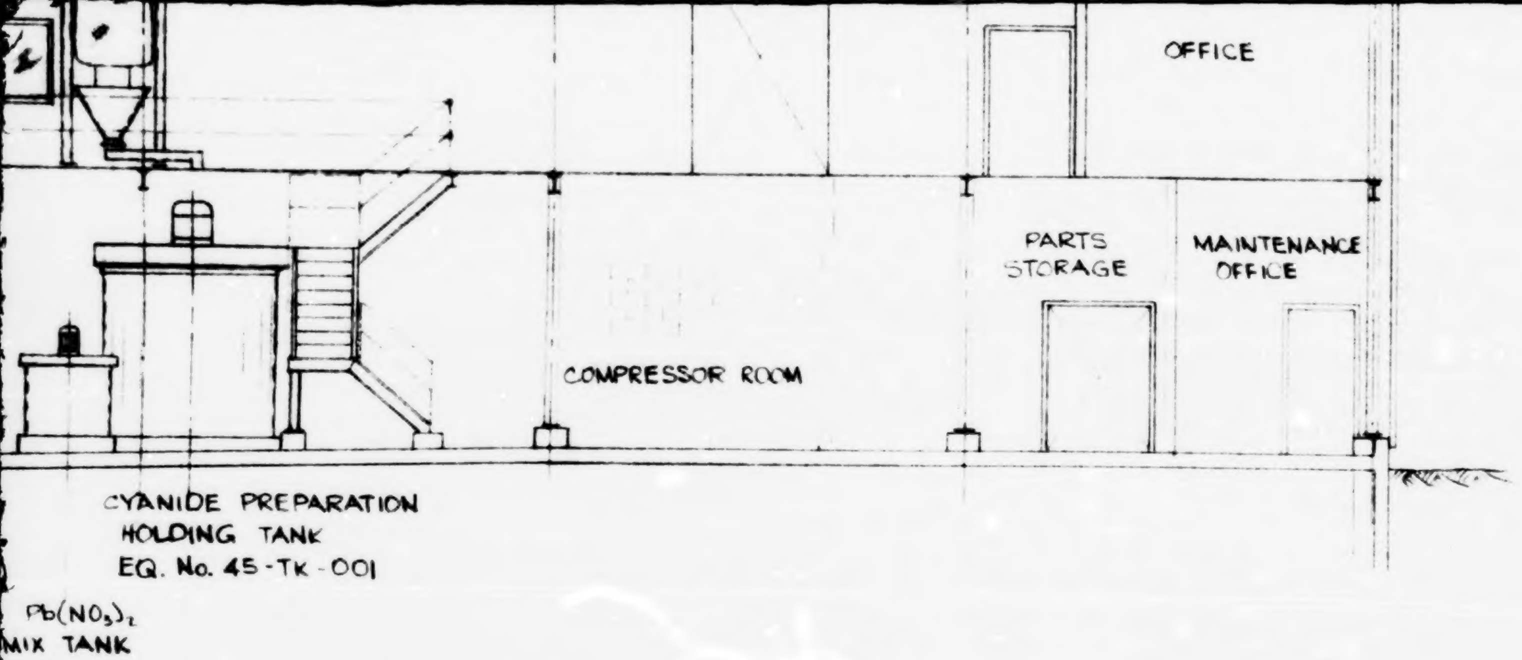

EQ. No. 43- FP- 001
-002
-003
ZINC FEEDER
EQ. No. 43- FR-001
EQ. 41-CC-001/002
DEAERATION TOWER
EQ. No. 42 DT-001
LIME MIX TANK
EQ. No. 45-TK-002
SUMP & PUMP.
EQ. No. 43-SP-001
Pb(N
MIX TA

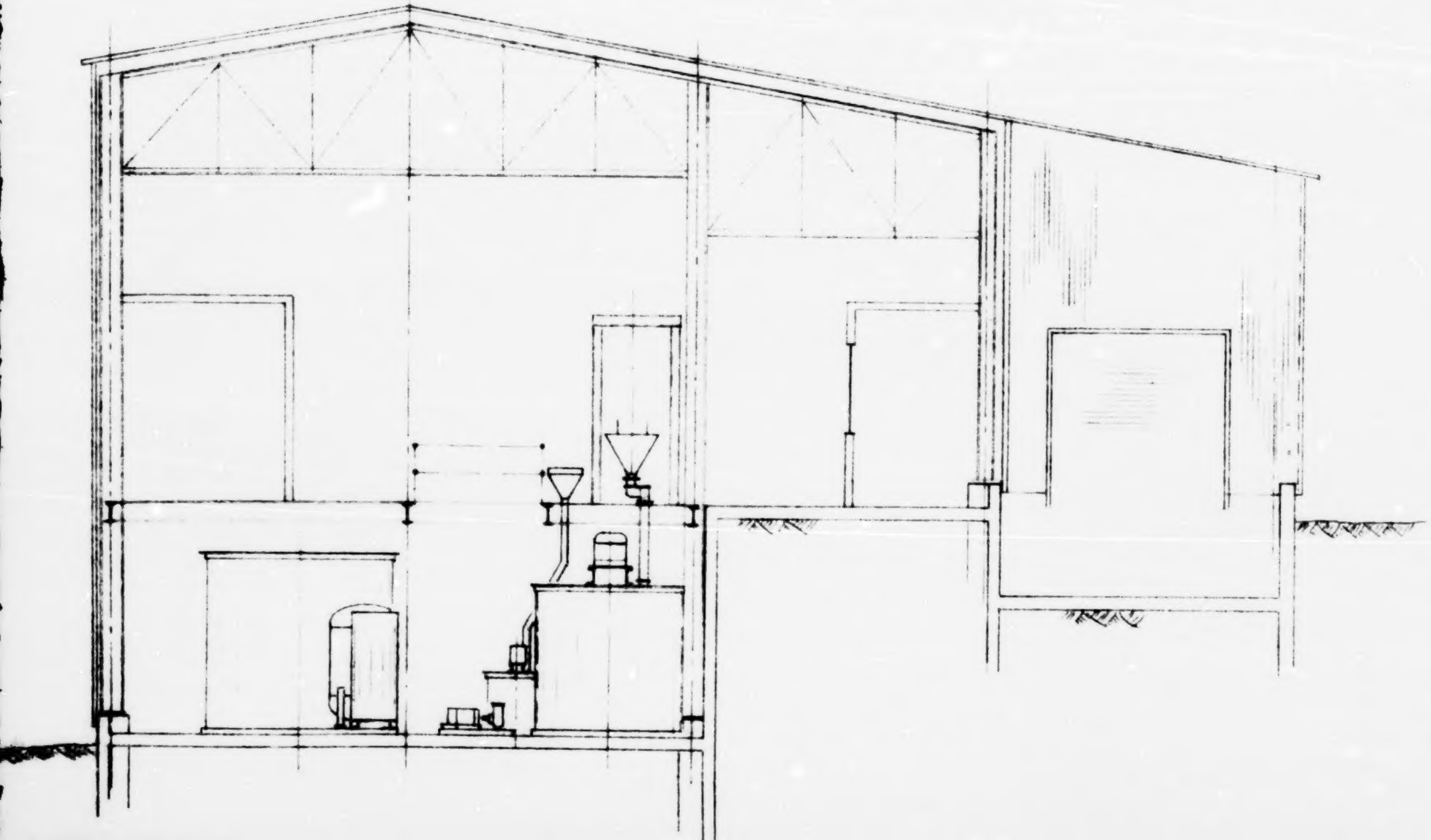


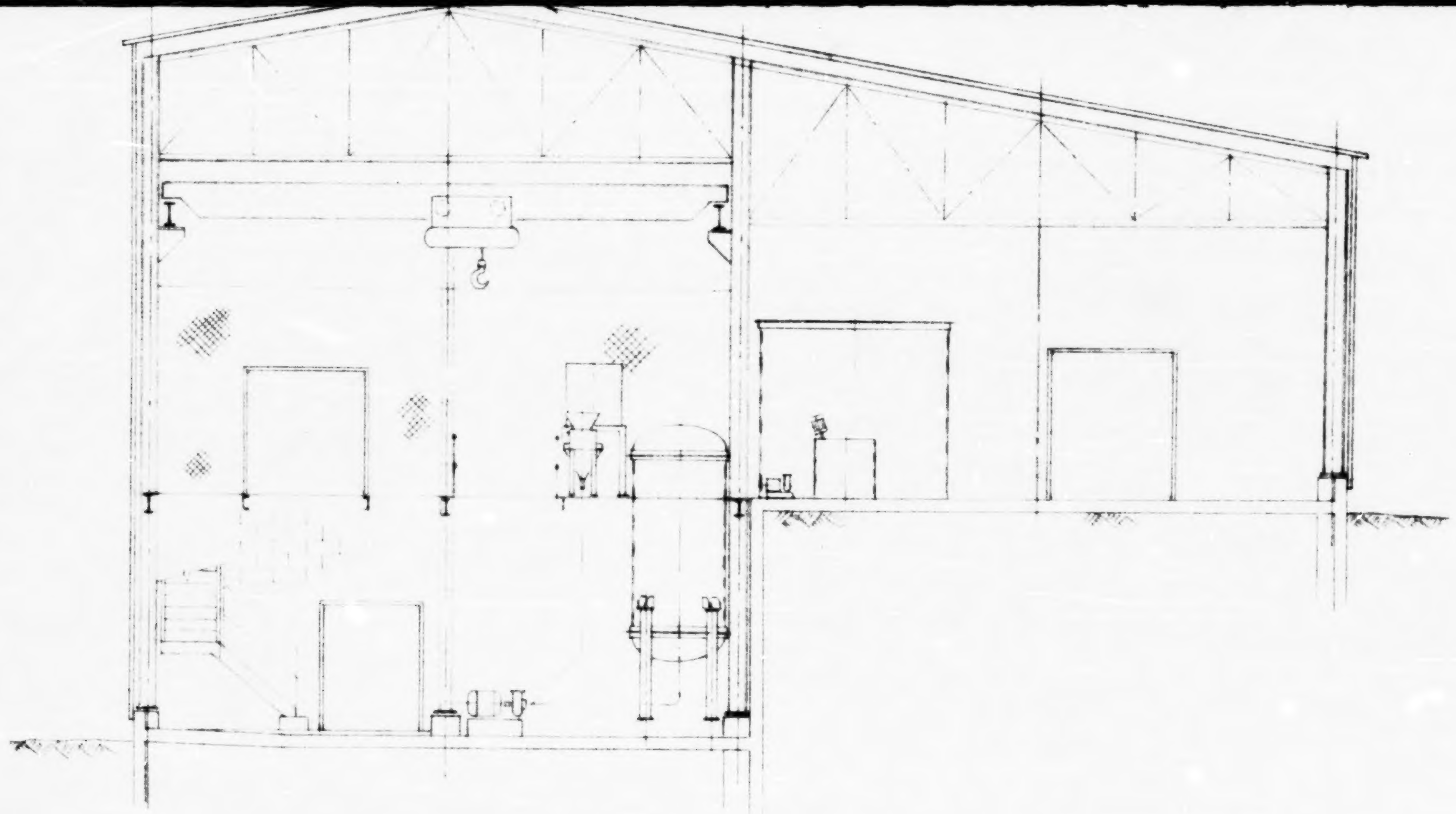

ZINC FEEDER
EQ. No. 43 · FR-001
OFFICE
DEAERATION
TOWER
EQ. No. 42 DT-001
PARTS
STORAGE
MAINTENANCE
OFFICE
COMPRESSOR ROOM
LIME MIX TANK
EQ. No. 45 · TK 002
CYANIDE PREPARATION
HOLDING TANK
EQ. No. 45 - TK - 001
SUMP & PUMP.
EQ. No. 43 - SP - 001
Pb(NO3)2
MIX TANK





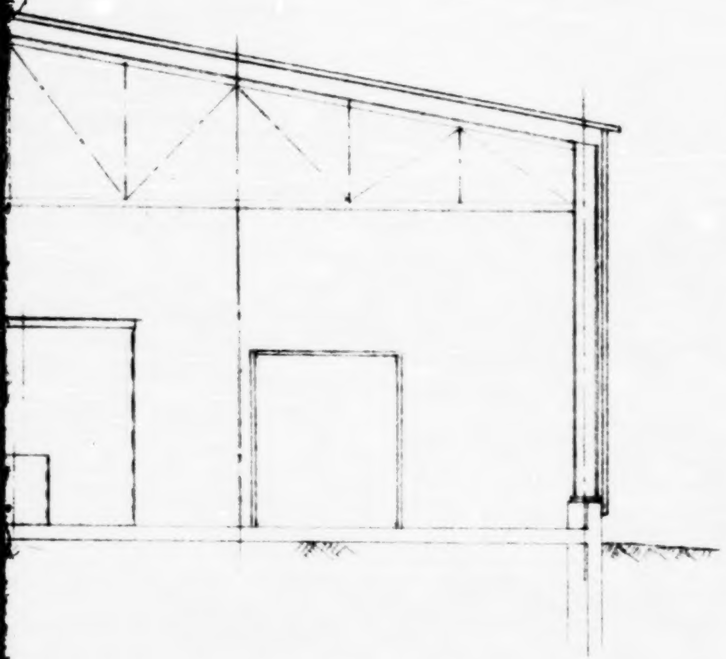

OFFICE
PARTS STORAGE
MAINTENANCE OFFICE
COMPRESSOR ROOM
CYANIDE PREPARATION HOLDING TANK EQ. No. 45-TK-001
$Pb(NO_3)_2$ MIX TANK

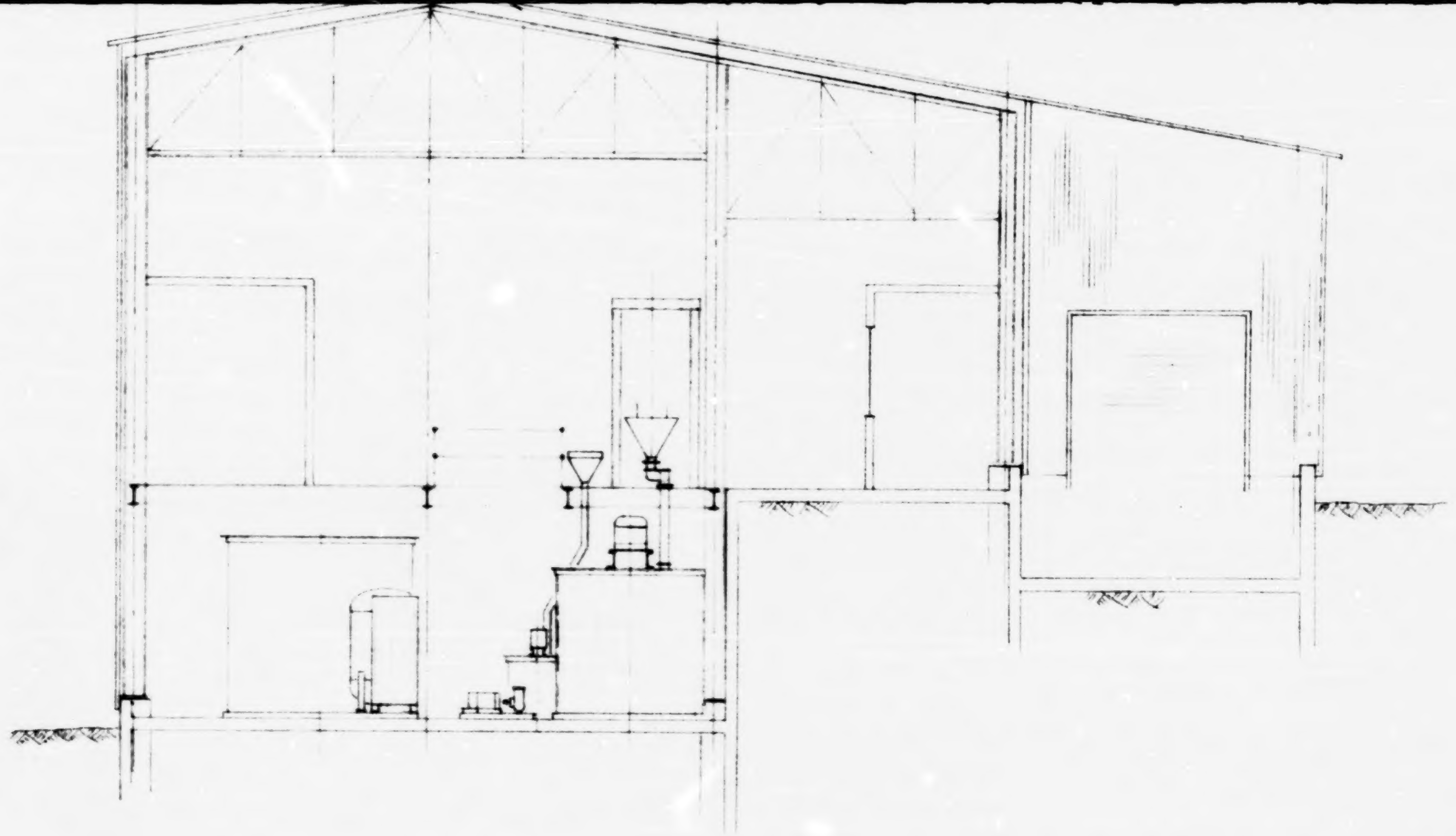




DWG. No.	REFERENCE	DWG. No.	REFERENCE	No.	DATE	REVISIONS	BY	CH

TRANSTEX





No.	DATE	REVISIONS	BY	CHECK	No.	DATE	REVISIONS	BY	CHECK
					P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
					P1		RELEASED FOR INFORMATION	E. N.	

SCALE	1:100	
DRAWN		16 JAN '87
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APPROVAL		

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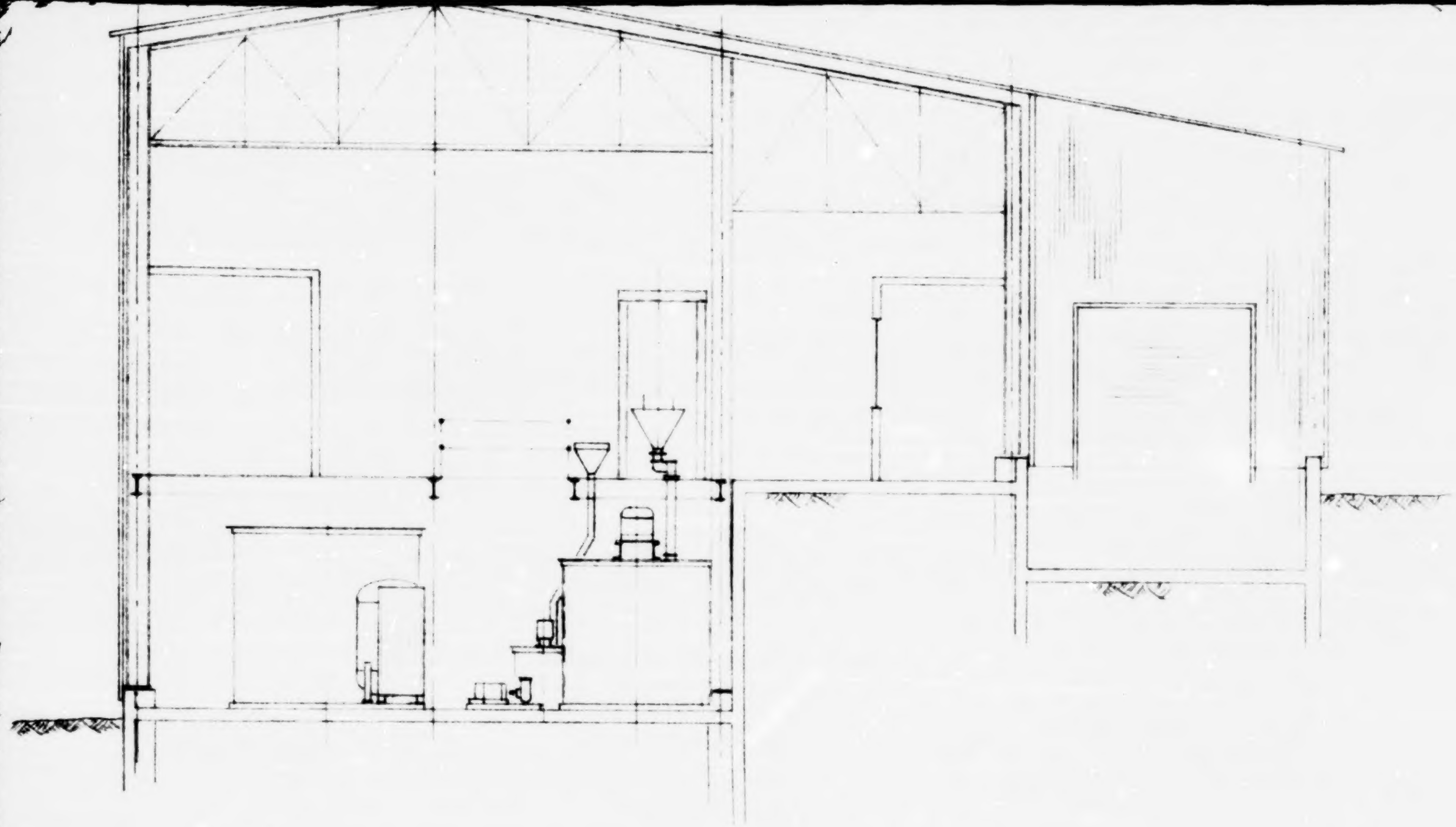
DATE



Y	CHECK	No.	DATE	REVISIONS	BY	CHECK
		P2	MAY 20,87	RELEASED FOR REPORT	J. LI	
		P1		RELEASED FOR INFORMATION	E.N.	

SCALE	1:100	
DRAWN		16 JAN '87
CHECKED		
APPROVAL		

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WITTECK DEVELOPMENT INC.

DSS - YUKON HEAP LEACH PROJECT
PROCESS PLANT (AREA 40)
PROCESS PLANT LAYOUT
SECTIONS

DATE	PROJECT NO.	DWG. No.	REV.
	5209	40-130-002	P2